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Cyanidation and Concentration
of
GOLD AND SILVER ORES

Lundberg Dorr and Wilson mill Ferry, South Dak^{ta} on F in Tail Gulch—the scene of many new developments in vanide technology
(Frontispiece)



Cyanidation and Concentration of GOLD AND SILVER ORES

BY

JOHN V. N. DORR, E.M., D.Sc.

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PREFACE

My purpose in writing this book is to record current cyanide practice throughout the world, giving only enough historical facts to serve as a background for the present development of the process and of the equipment used in its application. The inclusion of general testing procedure and numerous performance data is designed to insure maximum usefulness to the reader.

The fundamentals of mill practice and general types of equipment have changed little in the past fifteen years, although individual machines and general technique have been greatly improved. Outstanding developments have been confined principally to crushing, grinding and classification, adoption of flotation for certain types of ore and precipitation practice. A few of the older and well-designed plants not now in operation have been described because their practice was not far different from the best today, and because quite complete data on their operation were available. The description of plants now operating, both old and new, gives a fairly broad cross section of various practices since the general adoption of continuous methods.

I have approached this task with some hesitancy, which I am sure will be readily understood. But my relations with the metallurgical industry have extended over so many years and have received the cordial recognition of so many friends that I feel free to write as if I were still an operator, or consultant only.

My first introduction to cyanidation dates back 37 years when I was chemist, and later operator, under lease, at Deadwood, South Dakota, of one of the oldest cyanide mills in America, designed by the Gold and Silver Extraction Company about 1894. The first mill I built and operated, the Lundberg, Dorr and Wilson, at Terry, near by, was turned into a profitable undertaking by my invention of the Dorr classifier; and in remodeling another mill in the same district the Dorr thickener was born.

The Black Hills district of South Dakota, though small, presented many diverse metallurgical problems and yielded some

important developments. Out of it came the first crushing in cyanide solution in America, continuous decantation with mechanical thickeners, mechanical classification, continuous zinc-dust precipitation and the sluicing filter press of Merrill and the first successful use of Moore's vacuum filter.

So many have contributed to the development of the art of cyanidation that it is difficult, if not impossible, to make the record complete. I wish to express my thanks and appreciation to those managers and metallurgists all over the world who have published the results of their work and have so generously responded to requests for technical information; also to those other producers of the tools of the industry who have given their data so freely.

I am greatly indebted to my associates for assistance in the preparation of this work, especially to Mr. E. R. Ramsey, without whose aid it would never have been undertaken, and to Mr. Anthony Anable. Thanks are due also to Messrs. M. W. von Bernewitz, Burr A. Robinson and J. C. Williams for their aid in collecting and preparing the material for publication and to Messrs. H. A. Megraw and J. A. Baker for helpful comment and suggestions.

JOHN V. N. DORR.

NEW YORK CITY,
October, 1936.

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ABBREVIATIONS

In abstracting from the literature a certain style of abbreviation has been maintained, as follows:

Bulletin of Canadian Institution of Mining and Metallurgy (*Bul. C.I.M. and M.*), Montreal, Canada.

Bulletin of Institution of Mining and Metallurgy (*Bul. I.M. and M.*), London

Bulletin, Information Circular, Report of Investigation, or Technical Paper of United States Bureau of Mines (*Bul., I.C., R.I., T.P. U.S.B. of M.*).

Bulletin, Professional Paper, of United States Geological Survey (*Bul., P.P., U.S.G.S.*).

Canadian Mining Journal (*C.M.J.*), Gardendale, Quebec

Engineering and Mining Journal (*E. and M.J.*), New York City

Journal of Chemical, Metallurgical and Mining Society of South Africa (*Jour. C.M. and M.S.S.A.*), Johannesburg, Transvaal

Mining and Metallurgy (*M. and M.*), New York City

Mining and Scientific Press (*M. and S.P.*), San Francisco, California

Mining Magazine (*M.M.*), London, England.

Mining Journal (*M.J.*), London, England.

Proceedings, Australasian Institution of Mining and Metallurgy (*Proc., A.I.M. and M.*), Melbourne, Australia

Proceedings, Institution of Mining and Metallurgy (*Proc., I.M. and M.*), London, England

Transactions, American Institute of Mining and Metallurgical Engineers (*Trans., A.I.M.E.*), New York City.

NOTE: When the assay value of gold is expressed in terms of money the calculation is made at the price of \$20.67 per ounce unless otherwise stated

CYANIDATION AND CONCENTRATION OF GOLD AND SILVER ORES

CHAPTER I

HISTORICAL

The process that was to revolutionize all concepts of gold milling and increase a gold output that was even then lagging behind the needs of an expanding world-credit structure was developed without financial backing by three zealous experimenters in a poorly equipped laboratory in Glasgow, Scotland, almost half a century ago. To J. S. MacArthur, a metallurgical chemist, and to R. W. Forrest and W. Forrest, doctors of medicine, the cyanide process for gold extraction owes its origin.

At the time of their inadvertent discovery, MacArthur was chief chemist to the Tharsis Sulphur and Copper Co. in Glasgow, where he had experimented extensively on the recovery of small quantities of precious metals from copper leaching solutions and on increasing recovery by the use of chlorine and bromine. As this was outside his company's usual activities a small syndicate was formed, including MacArthur, the Doctors Forrest, and George Morton, a Glasgow business man. A room at the Forrests' surgery was fitted up as a laboratory. The results of some of his work were published by MacArthur in the *Journal of The Society of Chemical Industry* and as a result he was asked to investigate the Cassel process for the Cassel Gold Extraction Co., the inventor of the process, H. R. Cassel, having unexpectedly resigned. While MacArthur reported that the Cassel process for gold extraction, depending on the solvent action of electrolytically generated chlorine, could never become an economical process, he continued experiments in his laboratory on other

solvents, with the result that the dissolving action of a dilute solution of cyanide on gold was discovered.

Although L. Elsner, a German chemist, had published the findings of his experiments, which included the basic idea of cyaniding, in *Jour. f. prakt. Chem.* in 1846, he had failed to recognize its significance and made no practical use of it. It was MacArthur and the Forrests who realized the importance of their discoveries and fathered the idea through the vicissitudes of lengthy litigation, demonstration and finally commercial realization.

On Oct. 19, 1887, they registered their first patent—British Patent 14174. It covered the efficacy of potassium cyanide as a solvent in weak solutions. Their second patent, registered the following year, included the use of alkalis, methods of applying cyanide and the use of zinc "in a state of fine subdivision" for precipitation. A year later their extraction and precipitation inventions were patented in the United States by U. S. Patent 403202, granted on May 14, 1889. Practically without exception the patentees were unsuccessful in upholding their claims in the face of determined litigation in the Transvaal, New South Wales, Tasmania and finally in the United States.

The original claims of MacArthur and the Forrests are remarkable in that little has been disproved. Their claims with respect to the alkalinity of solutions (weak ones preferred) and to the use of zinc as a precipitant were basic. Nevertheless, the inventors were forced to modify their initial claim regarding the strength of solution, and also that in connection with the use of finely divided zinc as a precipitant was circumvented, in the early days, by the use of zinc filament or zinc shavings.

Following the growth of the cyanide process in the hands of other men, we trace its progress in the world's major gold fields—Australia, New Zealand, the Transvaal and the United States—for by 1888 in each of these areas British cyanide specialists were attracting attention. The first commercial cyanide plant was erected at the Crown mine, at Karangahake, New Zealand, in 1889, by J. McConnell. Alfred James, representing the MacArthur-Forrest company, introduced cyaniding on the Rand in 1890 at the Robinson mine. The first cyanide plants appeared in the United States in 1891, one at the Consoli-

dated Mercur in Utah, designed by H. W. Brown; and the other in Calumet, Calif., managed by A. B. Paul. Mexico's first cyanide plant was at El Oro mine, of the American Mining Company, in 1894. South African statistics show the rapidity with which the cyanide process was accepted in its infancy, for, from \$6,000 in 1890—the first year of cyanidation—the value credited to the process increased to \$6,000,000 in 1893.

In the early days in New Zealand and Australia the stamp battery was in the period of its greatest use, dry crushing hand-sorted ores. Crushing was followed by roasting and chlorination, since the ores were complex and not amenable to simple amalgamation.

With the advent of cyanidation the chlorination process, a costly system, declined and never revived. Later, as the metallurgy developed, weak cyanide solutions were used in the crushing systems instead of water.

Sand and slime were separated, first in pointed boxes or cones, the sand going directly to leaching vats, equipped with filter bottoms, where leaching baths were applied in succession, as required. The slime was settled in tanks and agitated in cyanide solution. Mechanical agitation was first used alone; then compressed air was introduced through tubes reaching to the bottom of the tanks and later by air lifts in tall cone-bottomed tanks—the Brown or Pachuca tank and the Parral tank, a modification of Brown's air-lift idea—and finally the Dorr agitator, combining air and mechanical agitation. Slime thickening and washing were accomplished in settling tanks, equipped for the decantation of clear, supernatant solution or in filter presses provided with facilities for washing the cake. Some ores were roasted before cyaniding, and others were treated successfully by the bromocyanide process.

Generally, fine zinc thread, or shaving, was used for precipitating the precious metal from cyanide solutions, employing upward percolation in the well-known baffle-type zinc box. Occasionally charcoal was used as a precipitant, and in Africa electrolytic precipitation on lead-foil cathodes.

The application of the cyanide process to ores, the value of which is principally or wholly in their silver content, lagged behind its use with gold ores. This is largely because silver usually occurred in combined form, sulphides and chlorides.

It was believed at first that these compounds would not yield to cyanide treatment.

About 1900, however, Leonard Holms made preliminary tests upon the silver sulphide ores of the Sirena mine at Guanajuato, Mexico, and soon thereafter E. M. Hamilton built and operated a large-scale experimental plant at the same property, definitely proving the usefulness of the process with ores of that character.

Previously, Hamilton had operated the plant built by Charles Butters, at Minas Prietas, Sonora, treating by cyanide a large quantity of gold-silver tailings accumulated from the operation of the Grand Central pan-amalgamation mill.

Butters and Mein in 1894 devised a revolving-arm distributor for uniformly charging vats, which made for economy and promoted more uniform leaching. In 1895 H. L. Sulman and F. L. Teed were working toward improved precipitation methods and probably were responsible for the stimulus that led up to the perfection of zinc-dust precipitation methods by C. W. Merrill at Marysville, Mont., and later at the Homestake.

Caldecott is credited with having developed agitation in 1898 from mechanical to air methods, starting the sequence that led up to the most modern methods. In April, 1901, Blaisdell patented his excavator for plowing leached sand to a central discharge outlet, whence it was carried by belt conveyor to waste. About 1899 L. H. Diehl introduced the tube mill, to accomplish finer grinding, borrowing the device from another industry.

While progress in cyanidation continued in Australia and the Transvaal, it was in the newer fields of Mexico, the United States and Canada that the greater number of improvements in the art had their genesis.

During my work in cyanidation in the western United States between 1899 and 1912, as chemist, as cyanide mill owner and operator, and finally as consulting engineer and plant designer, direct needs and an early recognition of the importance of continuous, foolproof operation, as well as the fundamental advantages of mechanical control of liquid-solid mixtures, led me to the invention of the Dorr classifier in 1904, the Dorr continuous thickener in 1905, and the Dorr agitator combining air and mechanical agitation in 1907, as well as to development of continuous countercurrent decantation. While each was pro-

duced to meet an individual need, all contributed materially toward higher recovery, lower costs and larger units by making finer grinding and a single all-sliming process more feasible. Many have contributed to improvements on these basic inventions. A. L. Blomfield, a former associate in a consulting capacity, developed a type of tray thickener that has been widely used and later the bowl-type classifier. Both machines were developed at the Golden Cycle mill, of which he was manager, to meet specific demands of mill expansion.

About 1903, George Moore, closely followed by Charles Butters, was applying the principle of vacuum-leaf filtration to cyaniding, and their filters came into wide use. The invention of E. L. Oliver, in 1907, of the segmental rotating vacuum filter made continuous filtration and washing possible and tended to supersede filters of the Moore and Butters types. Now the Oliver and other types of continuous filters are used in all parts of the world.

During the same period, continuous precipitation using zinc dust, excluding air from the system, was developed by C. W. Merrill, and this was perfected by T. B. Crowe, who applied vacuum to remove air from the solutions before precipitation.

Prior to the war period, 1914–1918, processes of cyanidation had become crystallized, the forerunners of present equipment had appeared in more or less developed forms and the industry was ready for consolidation of the advances. Thereafter, refinement of details received more attention, and the cyanide process entered its present stage. Gold production, which amounted to 21,303,725 oz. in 1914, was increased by discoveries, developments and new practices to a total of 27,474,516 oz. in 1934, the record production up to that time.

Multiple-stage crushing became standard practice, finer stages using rolls or cone crushers in closed circuit with screens, generally of the vibrating type. Larger jaw and gyratory crushers were used to handle larger pieces of ore; and fine crushers of improved design made possible a product of smaller size, $\frac{1}{4}$ in. or less. The stamp battery, still used in South Africa and a few mills in North America, was operated principally as a fine-crushing machine. Multiple-stage fine grinding is now used largely, the grinding machines in closed circuit with mechanical classifiers.

This latter period has brought emphasis upon the effort to reduce power costs and at the same time to grind finer than had been considered practically possible. The heavy-duty ball mill, short in proportion to its length, loaded with steel balls, has become of great importance. The advantage of high circulating loads between grinding mill and classifier led to the introduction of heavy-duty classifiers of both single stage and bowl type, and this trend is increasing. Many mills are producing a final pulp containing only a small percentage of plus 325-mesh solids.

In 1925 J. J. Denny, at the McIntyre Porcupine mill in Ontario, introduced differential grinding and selective agitation for the McIntyre ore, in which a large portion of the precious metals was locked in a refractory pyritic envelope. A bowl classifier at the end of the agitator series effected a concentration of the heavy pyrite, in the rake product, which was returned to the head of the mill for regrinding, while the less valuable quartz overflowed to decantation direct. Not only was the pyritic constituent ground finer than the quartz, but also it remained in the agitators much longer. Thus the quartz and pyrite were each ground and agitated to the degree demanded by their relative values and amenability, yet no uneconomical overgrind occurred. Much the same thing was done in South Africa at the Spring mines a little later. In the new McIntyre mill, built in 1931, Denny introduced flotation cells placed between ball mills and the classifier, to extract concentrates from the system as early as feasible.

Vacuum filters developed rapidly, and their use spread into all camps in the 1920's, especially those of the rotating-drum type of Oliver and, in lesser degree, the American disc-type filter of A. L. Genter. The first drum filter in Canada was used at the Hollinger mill for dewatering and washing the tailings from its countercurrent decantation plant.

At first, vacuum filters were used only for dewatering tailings or, at most, one stage of washing, in which case barren solution or water was sprayed on the cake to displace gold-bearing solution. Later, there was introduced a new system known as double filtration, wherein two filters with an intermediate repulping agitator were arranged in series for washing the pulp by a combination of spray washing and displacement. The question

of how many stages of decantation in thickeners and how many stages of vacuum filtration to use is dependent upon economic conditions, and while no definite trend is discernible, it appears that average practice favors three to four stages of decantation, followed by one stage of vacuum filtration. South African conditions are unique because Rand ore filters readily, and the slime can be washed easily by one stage of filtration. Intermittent filters of the Butters type were used to an appreciable extent, but new plants are now installing continuous rotating filters.

A study of cyanide methods indicates that no system has become more universally used than the Merrill-Crowe precipitation process. The use of zinc shavings has practically disappeared, except in the older mills. Clarification of pregnant solution in presses, deoxidation in vacuum towers (Crowe process), addition of zinc dust under conditions precluding the reabsorption of oxygen and, finally, the collection of the precipitate and excess zinc dust on filter presses or in filter bags have become standard practice.

Potassium cyanide was used generally in the early days of the cyanide process but was replaced later by the cheaper sodium cyanide. In 1917, owing to an acute shortage of white cyanide resulting from war conditions, black or "Aero" cyanide, $\text{Ca}(\text{CN})_2$, was introduced. Real del Monte, in Mexico, was the first to use the new cyanide in 1917, and such representative mines, as Golden Cycle, North Star, and Tonopah Extension rapidly followed the example of the Mexican mine. Today, black cyanide is extensively used in every mining district in the world.

Of great interest in precious-metal metallurgy is the flotation process long practiced in copper, lead and zinc concentrators. Its use in gold and silver milling really goes back only to about 1931, although countless laboratory and semicommercial scale tests antedated its use on a full-plant scale by more than a decade. Sometimes it is used alone for the recovery of gold and silver, yielding a concentrate which is shipped to a smelter and a tailing which is discarded, but usually it is employed in conjunction with cyaniding. Several options, in this latter regard, are available, such as flotation of the mill heads, followed by cyaniding the concentrates or cyaniding the heads with retreatment of the tails by flotation. Flotation, to be sure, is not a solution for all

milling problems, for its use is limited to certain ores, but it has become an increasingly valuable tool for the metallurgist.

The cyanide process in its development has made material contributions to many branches of technology. Not only in base-metal metallurgy but in nearly all industries where finely divided solids suspended in liquids are met it has been most valuable. Its recognition of the importance of mechanical control and continuous operation in fine solid-liquid mixtures and the size of its units have opened the way for advances in sewage and water purification and made wet chemical processes and industrial processes.

REVALUATION OF GOLD

The price of gold in the United States, formerly fixed at \$20.67 per ounce, began to rise Apr. 20, 1933, and continued to rise until it gradually attained \$30 per oz. during September of that same year. It was fixed at \$35 per oz. by presidential proclamation on Jan. 31, 1934, a price that was maintained through 1935 and to date, the first part of 1936. The price of gold in other world markets approximated the United States price range.

According to John J. Croston, *T.P. 709, A.I.M.E.*, preliminary estimates indicate that in 1935 somewhat over 30,000,000 oz. of gold was produced, the largest output ever recorded and exceeding the previous year by more than 11 per cent. Of this huge total the Witwatersrand field produced about 10,800,000 oz., compared with 10,480,000 oz. in 1934. Incomplete data indicate that the U.S.S.R. produced between five and five and a half million ounces, compared with 4,300,000 in 1934 and 2,814,000 in 1933. The United States took third place from Canada, with a production of 3,546,000 oz. against 2,916,000 in 1934, while Canada increased her total from 2,965,000 to 3,291,000 oz. Gains of lesser magnitude were made in Australia, the Gold Coast and other countries. As a result of great activity in developing outlying areas on the Rand, and the great number of new companies building mills in Canada, Australia, the Gold Coast and the United States, Croston prophesies that the years 1936 and 1937 should see new high records, as these new producers will more than offset any decline in the older properties.

HISTORICAL

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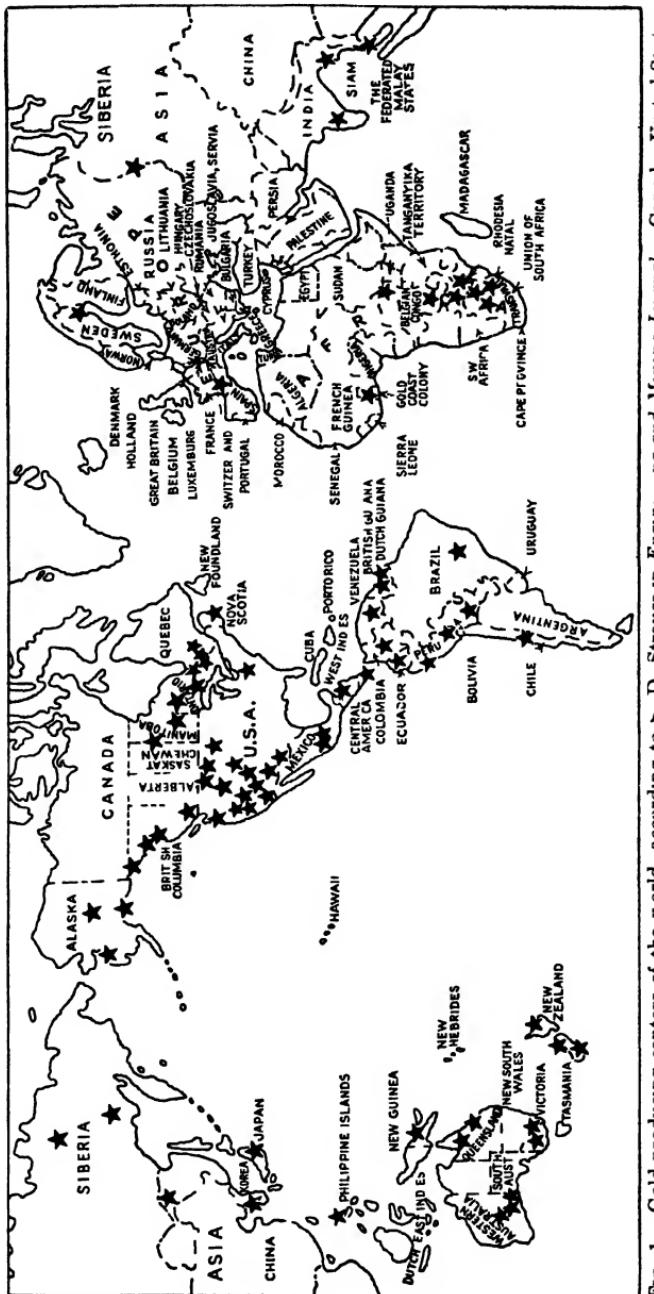


Fig. 1—Gold-producing centers of the world according to D. Strauss in *Engineering and Mining Journal*. Canada United States, and Mexico are the producers of most of the silver ores

Owing to the gradual depletion of known deposits that were profitable to work at costs existing prior to 1929, the world's gold-mining industry was slowly but steadily shrinking in importance. To quote Croston:

The International Geological Congress at Pretoria and the League of Nations Committee drew attention to the estimates of the mining experts of the various governments predicting a steady falling off in world production. On the Rand there were 10 mines that could not pay dividends in 1929, and the engineer of the Union Government predicted a drop to about 8,000,000 oz. annually by 1936 and to 4,000,000 oz. by 1945, based upon the costs existing at that time and upon the then standard price of 85 shillings. The United States and Australia, both producing at a relatively low rate, were slated for a still greater decline, and only Canada gave expectations of increased output. No account was taken at that time of Russian production, as the U.S.S.R. was not giving out information on the annual output or in regard to developments in that republic. It was known, however, that prior to the revolution most of the gold came from placers, and little was known about the possibility of large lodes.

The decline in production was based upon the inexorable fact that the deeper you dig and the leaner the ore treated the higher become the production costs. Several countries offered a bonus on gold production as a means of helping the struggling producers, and this in effect was the same as revaluing the gold as far as its sale by the mines was concerned.

If the world wants gold in sufficient needs for world commerce, it will have to pay a price commensurate with the cost of producing it. Gold increased almost four times in price during the period 1344 to 1717, when it was pegged at the equivalent of \$20.67 per troy oz. It would be ridiculous to attempt to prove that this price bore any relation to the cost of producing gold either in 1717, in 1890 or 1929. It was purely an arbitrary figure but with the discovery of rich deposits in California, Australia and South Africa was sufficient to bring forth a requisite volume of production. Methods of mining and treatment have reached a high degree of efficiency, and few radical reductions in per-ton costs are to be expected. The paucity of new discoveries and the necessity for treating lower grades from greater depths indicated a gradual drying up of production unless some stimulus were given to the industry.

This gradual shifting of the economic phases of world gold production has seemingly escaped the attention of the majority of monetary writers. They have treated gold production as a static affair unaffected by changes in reserves, grade, depth and other factors in exploiting this wasting natural asset. Certain phases of the economics of gold mining are not so simple as those of hog raising or wheat growing.

The first fillip to gold mining came with the onset of the depression. Labor became more abundant and efficient; wage rates dropped, as did the cost of supplies, enabling marginal producers to gain a better foothold. The second boost came when Great Britain went off the gold standard, enabling the producers in certain countries to benefit by a substantial exchange premium. The last and greatest step was taken when the United States Government officially revalued gold from \$20.67 to \$35 per oz. A careful consideration of the background factors leads to the conclusion that gold will not go back to its old price.

The new status that gold mining has attained has made it desirable to survey the industry throughout the world to ascertain how revaluation has affected production, ore reserves and costs.

The following tables were compiled by Croston from extensive files on all the important nonferrous metal mines of the world, to which clippings and annual reports from Johannesburg, London, Melbourne, Toronto, and other centers have been constantly added. They have been revised and brought up to date by him especially for this publication and are reproduced here with his permission.

ANNUAL OUTPUT OF WORLD'S LEADING GOLD MINES, 1929-1935
 (In fine ounces, troy)

Rank	Company	Location	Calendar years, unless otherwise noted					
			1935	1934	1933	1932	1931	1929
1	Crown		981,104	1,001,618	1,043,500	1,042,064	986,329	924,298
2	Government Areas	Transvaal, South Africa	857,919	915,851	1,033,687	1,146,140	1,129,872	856,005
3	Homestake	Transvaal, South Africa	744,743	730,936	821,334	775,313	652,606	1,086,163
4	Homedier	South Dakota, U.S.A.	546,515	471,749	510,368	479,188	451,860	595,249
5	East Rand Prop	Transvaal, South Africa	536,576	554,118	617,691	770,335	805,560	315,300
6	Sub. Nigel	Transvaal, South Africa	507,196	477,712	483,130	502,347	501,085	873,294
7	Lake Shore	Transvaal, South Africa	475,261	424,281	472,768	499,800	390,723	461,603
8	New State	Ontario, Canada	463,225 ^a	444,347	440,678	494,532	532,157	252,608
9	Springers	Transvaal, South Africa	435,245	424,266	434,925	481,279	438,954	293,587
10	Hollinger	Ontario, Canada	418,266	409,779	323,925	244,054	499,648	391,430
11	Dargafontein	Transvaal, South Africa	409,779	403,021	404,411	435,054	449,520	458,965
12	Brakpan	Transvaal, South Africa	360,924	331,492	292,209	224,603	56,003	455,094
13	East Geduld	Transvaal, South Africa	358,201	334,588	233,496	299,192	287,225	390,246
14	West Rand Cons.	Transvaal, South Africa	324,699	324,135	323,126	325,803	323,616	266,116
15	Geduld	Transvaal, South Africa	320,340	279,366	285,107	285,742	271,140	318,994
16	Consol. Main Reef	Transvaal, South Africa	302,688	301,017	319,274	337,696	334,457	263,591
17	Robinson Deep	Northern Sweden	264,187	238,948	128,805	56,900 ^b	27,760 ^c	253,147
18	Holden	Quebec, Canada	268,333	248,614	284,675	34,350	253,363	117,318
19	Noranda	Transvaal, South Africa	258,399	245,164	248,208	260,083	252,348	68,732
20	Simmer and Jack	Transvaal, South Africa	244,119	234,495	242,198	261,646	255,908	243,422
21	Modder East	Ontario, Canada	243,488	239,098	261,529	250,725	250,413	206,628
22	McIntyre Porcupine	Transvaal, South Africa	238,041	236,210	256,122	257,110	204,018	295,615
23	City Deep	Belgian Congo, South Africa	234,383	211,764	195,392	194,933	170,016	147,799
24	Kilo Moto	Transvaal, South Africa	230,774	220,416	246,400	261,855	298,940	128,766
25	Van Ryn Deep	Ontario, Canada	215,1028	218,203	177,190	171,299	297,602	294,909
26	Wright Hargreaves	Gold Coast, Africa	206,795	206,163	218,485	195,111	169,686	83,631
27	Dome	Transvaal, South Africa	199,653	190,797	177,143	175,063	169,360	173,464
28	Ashanti	Transvaal, South Africa	198,671	200,607	218,869	248,329	210,757	118,136
29	Langlaagte	Transvaal, South Africa	188,805	197,536	249,758	306,390	317,659	326,412
30	Modder "B"	Transvaal, South Africa	183,024	192,153	210,268	225,685	294,865	318,105
31	West Springs	Philippine Islands	180,441	173,514	205,876	225,481	210,498	218,054
32	Balatoc	Transvaal, South Africa	179,204 ^d	134,161	113,921	101,145	68,085	61,001
33	Witwatersrand Gold	Transvaal, South Africa	154,905	151,793	145,842	157,617	146,423	134,365

HISTORICAL

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36	Modder Deep	151,079	192,537	220,203	258,354	268,790	275,251	280,665
37	Durban Roodepoort Deep	149,979	151,774	169,287	188,671	183,020	172,362	164,809
38	Gelderhuis Deep	148,915	162,301	181,884	203,341	198,843	186,236	176,170
39	Tek-Hughes Deep	140,053 ^b	167,412	220,036	282,882	294,422	260,775	243,745
40	Lake View & Star/ ^f	133,811	152,885	170,177	93,476	109,261	132,378	118,568 ^a
41	Rose Deep	131,493	137,565	144,025 ^c	134,641	153,028	155,841	163,556
42	Bulolo (dredging)	124,359	106,061	90,375	52,982 ^d	124,782 ^y	135,192 ^y	135,388
43	New Kleinfontein	122,467	113,707	116,166	123,588	79,605 ^e	84,998	73,246
44	Bequia/ ^t	113,428	107,157	100,852	151,578	179,532	163,312	164,983
45	Alaska	118,908	128,015	150,967	97,917	103,800	113,800	113, ^x
46	Empire Star/ ^k	117,618	107,611	97,005	128,826	103,779	113,689	113,714
47	Witwatersrand Deep	118,805	116,442	129,921	119,779	112,339	128,656	140,330
48	Luijvard's Vie/ ⁱ	115,307	107,500	98,583	101,929	124,782 ^y	128,656	140,330
49	Can and Motor	111,459	107,451	111,875	119,235	128,746	90,120	80,120
50	Nundydroog	111,157	110,536	114,290	89,605	79,836	84,998	73,246
52*	Van Ryn Gold	108,501	128,438	124,531	123,085	58,988	119,024	112,870
53	Mysore, India	106,975	95,910	120,618	129,349	122,488	109,204	112,870
54	Minas Geraes, Brazil	94,771	92,507	91,324	88,598	102,488	101,904	102,228
55	Pioneer, Nigeria	102,267	96,857	105,849	109,109	115,473	123,161	108,228
56	Champion Reef	86,267	86,763	53,548	32,567	16,931	7,850	8,676
57	Yuba Cons./ ^j (dredging)	73,914	20,938	37 ^y	66,038	65,719	62,660	66,562
58	Natomas (dredging)	08,007	66,480	64,056	52,342	48,730	59,981	56,673
59	Siscoe	67,096	59,437	56,100	62,544	58,793	66,251	66,660
60	Belgian Congo, Africa	64,988	63,394	54,810	48,683	35,936	17,768	14,874
61	Grant Lacs Africaines (Alluvial)	59, ^k	51,024	49,577	43,018	36,910	31,315	32,441
62	South American Development	58, ^k	62,388	69,185	73,593	75,468	78,680	73,187
63	Martha (formerly Waihi)	56,124	59,938	65,204	75,092	65,865	65,961	60,310
64	Globe and Phoenix	55,929	54,004	28,626	24,325	25,231	12,488	11,700
65	New Goldfields of Venezuela	54,997	65,842	76,239	79,186	79,547	80,207	87,554
66	Great Boulder	54,989 ^b	50,337	44,607	39,919	43,437	38,303	33,168
67	Sylvania	54,789 ^b	50,337	44,607	39,919	43,437	38,303	33,168
68	Transvaal Gold	54,590	57,127	63,803	64,789	60,295	58,544	57,483
69	Oregum	52,944	50,398	51,311	52,223	63,083	63,700	62,483
70	Frontino/ ^t	49,970	32,232	32,275	46,649	37,130	18,492	15,939
71	Taquah and Abosso	49,408	49,452	44,453	39,888	42,922	44,946	41,046
72	Salsigne/ ^t	47,436	48,084	48,098	28,808	24,338	22,763	22,763
73	Idaho-Maryland	73,315 ^a	43,654	41,908	47,379	27,797	11,662	11,662
74	Bralorne	47,066	45,296	25,885	22,234	^y	^y	^y
75	Beattie	55,479	53,490	22,398	25,965	18,285	9,736	8,054
76	South American Gold and Platinum	44,846 ^m	28,134	25,965	25,965	18,285	9,048	8,054
77	Tao Gold	49,500	49,500	62,306	59,512	47,780	32,211	35,583
78	Sirau	43,000	46,500	46,153	54,168	54,702	48,455	48,455
79	Chosen (Korea)	38,528	40,116	39,547	58,010	39,349	32,900	29,453
80	Philippine Islands	31,024	17,980	6,156	12,794	12,794	34,124	34,124
	Colorado, U.S.A.	39,165	31,024	22,575	23,539	28,794	37,946	37,946
	United Gold Mines							

ANNUAL OUTPUT OF WORLD'S LEADING GOLD MINES, 1929-1935.—(Continued)

Rank	Company	Location	Calendar years, unless otherwise noted				
			1935	1934	1933	1932	1931
82	Oriental Cons. ^f	Chosen (Korea)	41,251	53,180	46,162	54,987	50,684
83	Wanderer	Southern Rhodesia, Africa	38,872	37,109	41,725	44,471	46,500
84	Sumitomo Konomai	Japan			42,865	28,935	4,381
85	Howey	Ontario, Canada	37,708	40,982	40,460	53,948	27,464
86	New Guinea Goldfields*	New Guinea	37,694	40,592	45,005	43,702	22,147
87	Mountain	California, U.S.A.	37,189	16,540	19,902	24,088	y
88	London	Colorado, U.S.A.					x
89	Itoion	Philippine Islands	35,767	31,764	33,500	41,672	y
90	Sons of Gwalia	Western Australia	34,261	42,740	38,500	42,100	38,225
91	Ariston*	Gold Coast, Africa	34,285	30,392	14,550	25,997	30,929
92	Mount Morgan ^f	Queensland, Australia	34,013	35,827	39,585	37,586	y
93	Cresson Cons.	Colorado, U.S.A.			28,400	26,365	32,276 ^a
94	Eastern Transvaal Cons.	Transvaal, South Africa			32,200	32,500	34,053 ^b
95	Capital (dredging)	California, U.S.A.			32,436	33,596	30,200
96	Conisaurum	Ontario, Canada	32,152	32,376	21,638	41,582	17,150
97	San Antonio	Manitoba, Canada	32,376	22,778	17,401	36,278	12,520
98	Premier	British Columbia, Canada	31,747	38,371	51,863	78,716	35,664
99	Glynn Lyttenburg	Transvaal, South Africa	31,633	33,085	33,167	32,394	30,641
100	Little Long Lac	Ontario, Canada	31,446	2,457	y	29,218	24,594
101	Golden Plateau ^f	Queensland, Australia	31,375	17,174	y	y	y
102	Kushikino	Kazashima, Japan			33,840	30,935	31,121
103	Nacasse	Ontario, Canada	32,585	32,056	3,725	y	29,818
104	Amalgamated Banket Areas	Gold Coast, Africa			y	y	29,642
105	Buffalo-Ankerite	Quebec, Canada	30,075	24,591	22,343	10,602	y
106	Lamaque	Ontario, Canada	29,123	y	y	y	42
107	Boulder Perseverance	Western Australia		33,053	45,000	45,541	42,101
108	Bibiani ^f	Gold Coast, Africa			3,201	45,500	52,818 ^c
109	Pickle Crow	Ontario, Canada	24,783	y	y	y	y
110	South Kalgurli	Western Australia	24,680	30,995	43,155	43,631	34,281
111	Yukon Cons. (Dredging)	Yukon, Canada	25,650	26,446	28,889	29,679	34,199
112	Kirkland Lake Gold	Ontario, Canada	22,215	22,315	18,465	25,323	29,180
113	Central Patricia	Ontario, Canada	22,061	6,373	y	28,315	25,764
114	North Kalgurli	Western Australia			23,215	23,487	16,899
115	Rediang Lebong	Netherlands East Indies			23,000	26,882	3,464
116	Reno	British Columbia, Canada	21,586	15,979	11,551	9,751	28,528
117	Original Sixteen-to-One	California, U.S.A.			20,858	26,678	1,790
118	Raub Australian ^d	Federated Malay States			26,136	24,727	23,382

23,382

18

34,040

18

119	Toburn	20,200	20,401	23,020	9,841	y	y	y	y
120	Young-Davidson	20,382	3,878	y	y	y	y	y	y
121	Botanario	y	19,851	22,913	y	y	y	y	y
122	Blackwater	y	14,966	20,813	22,649	26,595	25,650	15,160	y
123	Equatoria	y	20,055	20,723	15,456	19,205	16,502	2,254	y
124	Argonaut	19,722	19,722	21,357	27,086	23,762	32,300	34,957	x
125	Resende	19,538	19,538	18,307	26,691	31,348	1,241	y	y
126	Bellakor	16,800	16,800	10,225	4,475	1,267	y	y	y
127	Lava Cap	24,459	6,514	y	y	y	y	y	y
128	Lacefield	18,596	18,437	5,663	y	y	y	y	y
129	Northern Empire	17,758	17,758	20,261	22,186	22,964	27,450	35,746	24,970
130	Sherwood Starr	17,633	17,633	10,967	7,395	y	y	y	y
131	Cariboo	17,435	1,591	y	y	y	y	y	y
132	Island Mountain	17,078	2,961	y	y	y	y	y	y
133	McWatters	16,930	16,930	y	y	y	y	y	y
134	Carson Hill	16,739	2,021	y	y	y	y	y	y
135	Paymaster	16,712	16,546	y	y	y	y	y	y
136	Cuthbert's Misama [*]	y	y	y	y	y	y	y	y
137	Chen	y	y	y	y	y	y	y	y
138	Lucky Tiger	16,703	14,604	17,121	x	1,929	10,867	10,610	x
139	Tele	y	15,886	15,883	12,860	3,537	6,977	5,369	y
140	Lonely Reef	16,024	23,569	27,603	7,041	38,297	45,937	49,368	y
141	McKenzie Red Lake	15,183	y	y	y	y	y	y	y
142	Batuo	y	13,126	3,131	y	y	y	y	y
143	Rim (dredging)	y	14,965	14,673	14,512	8,131	9,840	11,500	y
144	New Pioneer Central Rand [/]	15,189	y	y	y	y	y	y	y
145	Cardinal	14,830	y	14,704	x	x	x	x	x
146	Bellire	y	10,923	y	y	y	y	y	y
147	Panique	y	18,809	y	y	y	y	y	y
148	Kennedy	y	11,866	y	y	y	y	y	y
149	Ipo	12,483	19,660	27,043	41,742	48,207	30,541	14,756	y
150	Eureka-Standard	14,527	x	30,460	11,305	25,439	27,483	30,565	y
151	Pato Cons. (dredging)	7,645	4,793	y	y	y	y	y	y
152	Central Eureka [*]	13,661	13,605	10,337	8,034	y	y	13,575	23,679
153	Maree [*]	13,432	4,462	y	y	y	y	y	y
154	Sullivan Cons.	12,881	12,881	12,784	13,094	14,012	10,778	21,181	y
155	Rhodesian Corp., "Fred"	12,451	10,666	y	y	y	y	y	y
156	Norseman [*]	12,451	12,451	10,666	17,313	5,114	y	y	y
157	Ashley	12,358	13,181	y	y	y	y	y	y
158	Dentonia	12,027	4,380	y	y	y	y	y	y
159	Aniago-Huronian	y	15,541	24,245	21,939	43,883	39,569	24,654	x
160	Central Manitoba [*]	11,970	11,659	12,108	19,447	19,951	19,781	20,379	x
161	Golden Valley	11,586	y	y	y	y	y	y	y
162	Rietfontein Cons.	11,100	y	y	y	y	y	y	y
163	Phoenix Prince	10,819	y	y	y	y	y	y	y

ANNUAL OUTPUT OF WORLD'S LEADING GOLD MINES, 1929-1935.—(Continued)

Rank	Company	Location	Calendar years, unless otherwise noted					
			1935	1934	1933	1932	1931	1930
164	Zuiho	Taiwan, Japan						
165	Manitobewan Cons.	Ontario, Canada	10,113	679				
166	Mount Coolon	Queensland, Australia		32,820	38,444			
167	Verguas	Panama	5,064	14,979	y	5,951	y	9,060
						10,144	y	7,835
						y	y	y
						y	y	y

^a Local government statistics.^b Local government statistics.^c Estimated from government figures.^d Mill burned down Oct. 29, 1929.^e Year ended Sept. 30.^f Year ended June 30.^g Sixteen months ended June 30.^h Unofficial estimates.ⁱ Year ended March 31.^j Year ended Feb. 28.^k No data released. Approximate importance shown by relative rank.^l Crude gold.^m Year ended Aug. 31.ⁿ Fifteen months ended Dec. 31.^o Shutdown. Mill destroyed by fire.^p Eleven months to April 30.^q Year ended April 30.^r Year ended July 31.^s Year ended Oct. 31.

* The 53 mines of over 100,000 oz. annual output include the dredging enterprises of the U.S. Smelting, Refining and Mining Company in Alaska. Their production is not given separately but included in total of that company.
 x No data received.
 y Mine not in operation or not yet discovered.

**GOLD PRODUCTION OF CUSTOM SMELTING, SILVER AND BASE-METAL
MINING COMPANIES**

(1935 figures included where reports are available)

Company	1935	1934	1933	1932	1931	1930	1929
Amer. Smelt. & Ref. Co.	1,969,299	1,476,487	1,208,697	1,345,960	1,454,774	1,700,702	1,461,215
Amer. Metal Co. Ltd.	578,960	414,284	335,371	321,535	522,902	485,176	478,504
U.S. Smelt., Ref. & Min. Co.	356,966	321,385	328,826	350,678	314,755	322,968	266,197
Golden Cycle Corp.	158,941	162,598	145,271	158,522	122,711	x	x
Hudson Bay M. & S. Co. Ltd.	104,218	99,334	94,745	82,565	73,000	y	y
Internat. Nickel Co. ^a	89,944	74,375	21,335	22,675	23,381	22,867	7,802
Hidachi mine (Nippon Mining Co.)				87,996	81,953	84,063	79,753
Saganoseki mine (Nippon Mining Co.)				72,764	68,514	122,890	98,333
Philip Dodge Corp.	107,435	68,880	43,882	28,736	99,749	70,033 ^a	100,607 ^a
Cia. Minera las dos Estrellas	67,045	44,069	39,264	25,309	80,153	78,417	62,920
Utah Copper Co.					64,124	64,240	116,087
San Luis Mining Co.					46,555	43,919	40,800
Nachima mine (Mitsu- bishi Mining Co.)			41,874	39,262	36,166	30,805	23,632
Cons. Min. & Smelt. Co. Can.	65,131	35,328	22,393	33,346	24,168	25,782	14,694
Besshi (Sumitomo Min- ing Co.)			26,587	23,567	38,395	28,662	24,467
Cerro de Pasco Copper Corp.	28,637	20,000	20,326	12,000	28,070	29,778	33,671
Cia. Dos Carlos, S.A.					24,403	31,090	29,354
Cia. Min. y Beneficiadora de Inde					22,442	25,500	16,691
Greene Camunea Copper Co.	16,370	20,851	13,528	9,596	8,447	7,941	12,366
Kosaka mine (Fujita Mining Co.)			24,236	17,355	18,558	18,349	17,500
Anaconda Copper Min- ing Co.	49,208	40,886	18,551 ^c	32,724	21,209 ^c	61,980	
Mexican Corp. ^b	20,460	13,193	6,744	8,339	9,195	7,941	9,838
Shenandoah-Dives Min- ing Co.		15,969	23,124	x	30,561	17,753	x
Howe Sound Co. ^d	10,690	12,034	13,701	8,868	5,476	12,770	14,252
Neg. Min. Santa Marin de la Paz	12,787	x	x	x	8,874	6,269	12,025
New York & Honduras Rosario Mining Co.	12,274	12,996	17,210	16,054	15,835	13,498	10,304
Guanajuato Red. & Mines Co.		11,810	12,482	15,065	16,526	11,382	12,346
Magma Copper Co.	32,890	30,311	11,255 ^c	8,120 ^c	9,244 ^c	10,154 ^c	13,405 ^c
Union Min. du Haut Katanga		20,255	9,391	x	x	x	x
Bunker Hill & Sullivan Min. & Cone. Co.	5,230 ^c	11,101 ^c	34,760	44,811	46,000	x	
Andes Copper Mining Co.	13,351	10,459	6,026	1,991	4,791	6,629	16,537
United Verde Copper Co.		shut	shut	shut	10,350	42,937	62,097
Nevada Cons. Copper Co.				8,311	26,670	14,537	14,536

^a Calumet and Arizona output included, although not merged during these years.

^b Year ended June 30. Cyanide plant operations only.

^c Own ores only, custom ores not included.

^d Annual sales.

^e No data received.

^f Mine not in operation or not yet discovered.

* The gold production of the Société Française des Mines de Bor (Yugoslavia) is not included in the above table as the figures are not available. It is believed that about 2 tons. are produced for each ton of copper metal—or about 60,000-75,000 tons. annually in recent years.

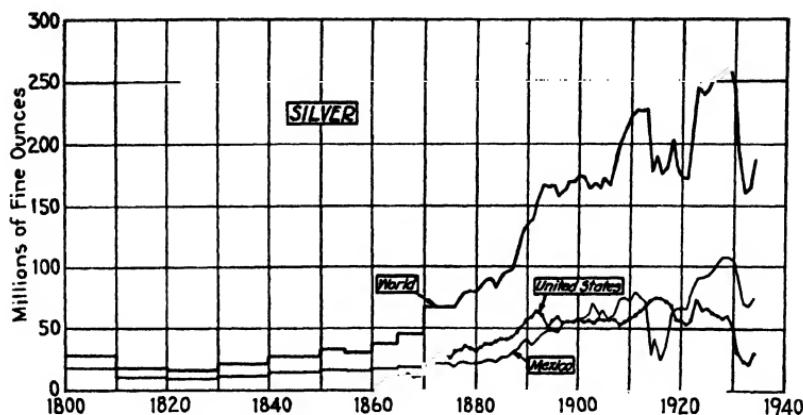


FIG. 2A.—“The Best Year for Gold and the Poorest for Silver” by Scott Turner, Director of U. S. Bureau of Mines. (*From Mining and Metallurgy, January, 1933.*)—Silver.

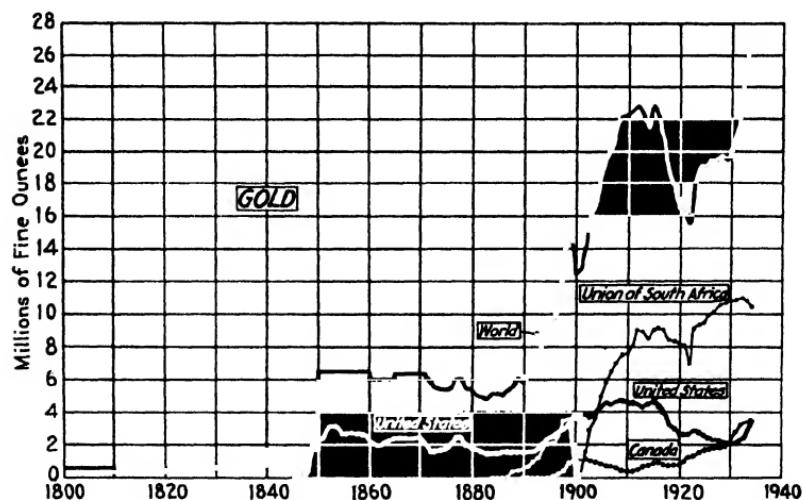


FIG. 2B.—“The Best Year for Gold and the Poorest for Silver,” by Scott Turner, Director of U. S. Bureau of Mines. (*From Mining and Metallurgy, January, 1933.*)—Gold.

GOLD PRODUCTION OF THE WORLD, BY COUNTRIES
 (In troy ounces, fine gold)

	1931	1932	1933	1934 ^a
United States (inc. Philippines).....	2,395,722	2,449,032	2,550,246	3,075,102
Canada.....	2,695,219	3,050,581	2,949,309	2,904,395
Mexico.....	623,003	584,487	637,727	670,000
Total North America.....	5,713,944	6,084,100	6,143,282	6,709,587
Total Central America.....	67,725	82,238	87,075	97,000
Total South America.....	579,134	685,981	909,359	1,109,000
Transvaal.....	10,877,777	11,558,632	11,013,713	10,486,303
Rhodesia.....	541,505	580,484	645,687	693,263
West Africa.....	259,023	292,510	338,110	377,100
Congo, Madagascar, etc.	249,656	304,453	430,302	510,900
Total Africa.....	11,927,961	12,735,979	12,427,212	12,007,656
U.S.S.R. (inc. Siberia).....	1,700,960 ^b	1,990,085 ^b	2,667,100 ^b	4,313,000 ^b
Other Europe.....	259,346	298,145	405,931	506,000
Total Europe.....	1,960,306	2,288,230	3,073,031	4,819,000
British India.....	330,484	329,632	336,106	320,563
East Indies.....	100,083	77,964	78,832	89,100
Japan and Chosen.....	642,863	642,663	642,426	700,000
China and others.....	223,291	225,707	300,339	350,000
Total Asia.....	1,296,521	1,275,966	1,357,703	1,459,663
Australia and New Zealand.....	783,934	998,267	1,152,471	1,212,610
Total for world, ounces.....	22,329,525	24,150,761	25,150,133	27,474,516
Total value old ^c	\$461,591,475	\$499,239,701	\$519,898,519	\$507,949,700
Total value new ^d	\$781,533,375	\$845,276,635	\$880,254,655	\$961,608,060

^a Preliminary estimates.^b Unofficial data based on Soviet press reports; 1934 includes 50,000 oz. from ores exported to United States for recovery.^c Value: \$20.6718 + per fine oz.^d Value: \$35 per fine oz.From *Mining and Metallurgy*, January, 1933, article by Scott Turner, Director of U. S. Bureau of Mines.

SUMMARY OF WORLD'S REPORTED RESERVES OF GOLD ORE^a

Region	After revaluation			Before revaluation				
	Tonnage all companies (including those not reporting grade)	Tonnage (companies reporting grade only)	Total metallic ounces fine gold	Average grade	Tonnage all companies (including those not reporting grade)	Tonnage (companies reporting grade only)	Total metallic ounces fine Gold	Average grade
Transvaal, South Africa.....	187,830,094	183,580,964	50,481,838	0.2748	101,034,871	35,484,737	0,3512	
Rhodian mines.....	^b	174,667,878	48,155,334	0.2773	^b	1,904,100	936,421	0,4918
Other African.....	^b	3,315,334	1,211,558	0.3673	^b	1,818,065	1,317,242	0,7345
Australasian (not including Mount Morgan).....	18,923,449	18,723,449	3,742,264	0.4194	^b	12,821,812	3,745,442	0,2921
Philippines.....	^b	5,892,913	0,3147	^b				
Asia.....	11,270,919	11,070,919	4,262,924	0.3851	^b	4,821,812	1,997,442	0,4143
South American.....	^b	2,608,888	1,112,115	0.4263		992,096	714,451	0,7201
United States.....	^b	2,293,835	1,190,933	0,4843	^b	1,169,621	665,598	0,5691
Canada (not including Noranda).....	56,948,266	2,644,200	1,392,713	0,5267	^b	2,117,097	1,094,242	0,5169
Canada.....	63,381,313	2,180,425	1,610,927	0,558				
Noranda.....	32,532,313	61,004,996	14,356,536	0,2335	^b	37,708,286	8,745,833	0,2552
Europe.....	5,900,000	30,012,996	8,950,563	0,2982	^b	15,258,286	11,827,136	0,3887
Alluvial mines.....		1,408,000 ^c	1,538,600					
Dredging and hydraulic mining but not yardage).....		1,175,981,112	8,429,242	25 ^d				
World total (by classes)			1,910,105				1,894,264	
Ores.....	3,53,004,800	285,311,581	80,117,837	0.2808		156,134,798	53,986,957	
Alluvial mines.....	^b	1,408,000	1,538,600					
Dredging and hydraulic mining.....	1,175,981,112	8,429,242	25 ^d					
Dredging (no yardage reported).....			1,910,105				1,894,264	
World grand total.....			91,995,784				55,881,221	

^a Not including Russia.^b All companies listed reported grade.^c U.S. data not complete.^d Square fathoms.^e Cubic yards.^f Cents per cubic yard.

SILVER PRODUCTION OF THE WORLD, BY COUNTRIES
 (In troy ounces, fine silver)

	1931	1932	1933	1934*
United States (inc. Philippines)	30,919,135	23,980,773	23,002,629	31,384,218
Canada	20,558,216	18,356,393	15,187,063	16,380,029
Mexico	86,064,457	69,303,054	68,101,062	75,000,000
Central America	4,000,000	4,300,000	4,800,000	5,000,000
South America	17,200,004	11,190,661	11,272,860	15,890,000
Total America	158,741,812	127,130,881	122,363,614	143,624,247
British India (inc. Burma)	5,923,005	6,026,737	6,080,241	6,120,000
Japan and Chosen	6,386,777	6,569,975	6,168,174	6,300,000
China and others	482,320	280,437	80,437	200,000
East Indies	1,473,113	842,362	860,463	900,000
Total Asia	14,061,721	13,719,511	13,189,315	13,520,000
Transvaal	1,063,050	1,120,668	1,075,011	1,002,000
Rhodesia	76,508	114,803	112,459	120,000
Belgian Congo	2,411,250 ^b	1,864,700 ^b	2,320,225 ^b	3,858,000 ^b
Other Africa	1,222,346	340,000	381,000	231,000
Total Africa	4,773,154	3,440,261	4,178,695	5,211,000
Europe	12,927,567	13,144,462	12,433,895	13,500,000
Australia and New Zealand	8,628,744	9,492,726	11,552,438	12,000,000
Total output in ounces	199,132,998	166,927,841	163,717,957	187,855,247
Total value in dollars	57,774,457	47,080,328	57,296,373	90,702,149
Average price in cents	29.013	28.204	34.997	48.283

* Estimated.

^b American Bureau of Metal Statistics.

Figures for foreign countries are based largely on the statistics collected by the Bureau of the Mint where no direct official figures can be obtained.

From *Mining and Metallurgy*, January, 1933, article by Scott Turner, director of U.S. Bureau of Mines.

CHAPTER II

EXAMINATION AND TESTING OF ORE

Reputable firms and experienced metallurgists who specialize in the testing of ores are to be found in all of the principal mining countries. Laboratories operated by governmental agencies conduct preliminary tests on ores for prospector or mineowner in Australia and Canada.

Leading manufacturers of machinery and chemicals used in cyanide and flotation plants maintain testing plants where the essential factors governing the proper use of their products are determined by experienced experimenters.

For all those properly qualified by education or experience, who desire to carry out their own tests, a description of the general methods of conducting them and of the apparatus required is outlined in this chapter.

PRELIMINARIES

The value of any series of experiments depends entirely upon the validity of the sample of ore used for test. It must represent, wholly and completely, the character of the material that will be treated in the plant to be designed and constructed. If it does not, the tests will be useless, or worse—misleading.

In some cases, owing to conditions under which mining is carried on, it is considered advisable to remove a portion of valueless material by sorting before the ore is delivered to the metallurgical plant. No intelligent appraisal of the need for sorting or its extent can be determined from a laboratory sample, but it is essential that the test sample should represent accurately the character of the proposed mill feed, as to both precious-metal content and the proportions of barren material, wall rock, gangue, etc. All of these factors are important, since they go to determine the sizes of equipment for settling, filtering, clarifying and classifying as well as grinding equipment. These details are crucial and are essential parts of a complete investigation of possible metallurgical processes. Therefore, if sorting is to be done, the test sample should represent the ore after sorting.

The quantity of the sample is important in its bearing on the representative nature of the material. It should be sufficient

to represent truly the character of the ore. Canadian custom at present is to ship from 500 lb. to 1 ton for Ottawa tests. Where the material is unusually uniform, smaller amounts will serve as well.

In cutting down the sample and making separate portions for different methods of treatment, all possible care should be taken to ensure similarity of each portion. No care is too great to make certain of the representative character of the sample for testing. Consideration should be given also to abnormalities of ore feed that may be expected, owing to variations of the ore bodies being mined, to alter the average or usual character of the mill feed.

MICROSCOPY AS AN AID IN CYANIDATION PROBLEMS

The assistance that can be obtained from the microscope in solving ore-dressing problems has been increasingly appreciated in the past ten years, as attested by the frequency of papers on the subject.

Observations by means of binocular microscopes with magnifications up to 100 diameters and corresponding resolutions have long been common practice, and such observations are a great help in working out methods for the satisfactory treatment of an ore.

Further progress resulted in the use of metallographic equipment and magnifications up to 500 diameters. However, it is only recently that full use has been made of metallographic microscopes with useful magnifications up to 1000 diameters wherein particles 0.5 micron in diameter or less are clearly resolved. Such work requires skillful preparation of the specimen before the full capacity of microscope can be made use of.

The latter technique is of especial value in connection with cyanidation of gold and silver ores, because in certain ores some of the gold occurs as minute blebs or stringers in pyrite, the inclusions often being as fine as 1 micron in diameter. Two excellent examples of such occurrences published recently are (1) "Increasing Gold Recovery from Noranda's Milling Ore," by C. G. McLachlan, *Trans. 112, A.I.M.E.*, 570, 1935; and (2) *The Role of the Microscope in Ore Dressing*, American Cyanamid Company, 1935, pp. 12-14.

By permission of American Cyanamid Company the following illustrative example is quoted from the paper referred to above:

Typical Problems—Example I.

A sample of auriferous-pyrite concentrate assaying 4.75 oz. ton was submitted to our ore-dressing laboratory for the purpose of determining whether the gold could be extracted by means of straight cyanidation.

Preliminary tests on this sample indicated that this material was very refractory. Subsequent tests, in which the raw concentrate was reground to -325 mesh and cyanided for a long period of time with a



FIG. 3.—Photomicrographs showing tiny inclusions of gold (G) in pyrite (Py). Note that the gold (G) is in very minute white specks, surrounded by a black area which is gangue.

A. Magnification 1000 \times . Note in this large piece of pyrite there is one piece of gold that is less than 1 micron. In neither illustration is there gold as large as three microns in diameter. B. Magnification 1000 \times .

strong solution, failed to improve the extraction materially. The lowest cyanide residue contained 0.33 oz. per ton. At this stage in the investigation it was decided to submit a sample of the above residue to the microscopical laboratory for the purpose of determining the form and manner of association of the gold. Accordingly, samples of the +325-mesh and -325-mesh residues were briquetted, polished, and examined with the metallograph at high magnification. . . . Figures 3A and 3B are photomicrographs of this residue at a magnification of 1000 diameters. It will be noted that the gold occurs as metallic gold completely encased in pyrite and that the size of the gold particles is about 1 to 3 microns.

Inasmuch as -325 mesh is about the present economic limit for grinding, it was useless to proceed with further tests along the lines of finer grinding and cyanidation of the raw concentrate. Thus at an early stage in the investigation, the intelligent use of the microscope saved much useless work by narrowing down the lines of attack and pointing to a practical solution of the problem. In this case either roasting prior to cyanidation or direct smelting of the concentrate was definitely indicated.

For further information on the subject, the reader is referred to the latter paper which gives an excellent description of a modern microscopic laboratory for such ore-dressing investigations, with practical examples, methods used and bibliography.

A recent paper by R. E. Head¹ presented at the February, 1936, meeting of the A.I.M. and M. Engineers, subject, "Physical Characteristics of Gold Lost in Tailings," indicates the possibilities of the microscope in the study of tailings losses. The following is an extract:

SURFACE CONTAMINATION

The experience gained by microscopic study of numerous gold ores and tailings has shown that there are pronounced differences in the physical characteristics of the minerals composing them, more especially of the gold. Repeated studies of tailings from flotation and cyanidation of gold ores has established the fact that surface contaminations on gold particles are often directly responsible for high gold losses; comparison of gold particles isolated from flotation concentrates and the resulting tailings have showed that the clean gold has been recovered and the tarnished or contaminated gold invariably lost in the tailing. Obviously, it is not possible to make such a comparison of gold ores treated by cyanidation, as the clean gold has been taken into solution; but when the gold found in cyanide tailing has a tarnished or coated surface, one may infer, with a reasonable degree of certainty, that the clean gold has been extracted. This premise is supported by experimental evidence obtained by isolating particles of tarnished gold and exposing them to cyanide solution in small parting cups. In one such experiment, tarnished gold particles picked from a cyanide tailing showed but slight evidence of dissolving at the end of 27 days. In this test, a cyanide solution of 1.6 lb. per ton was used, and the leach solution was decanted and renewed every 24 hr. The proof of cyanide attack was manifested

¹ Microscopist, Intermountain Experiment Station, U. S. Bureau of Mines, Salt Lake City, Utah.

by a noticeable thinning of the gold particles at the edges. A rim of a substance that appeared to be gelatinous was visible at the edges of the gold particles, and there is reason to believe that this material encased the entire surface of the gold particles. It is not known whether the gelatinous film is of secondary origin, resulting from a reaction between the cyanide and some substance or substances adhering to the gold surfaces in the form of a coating, or is an original constituent of the surface contamination which has been made visible through the dissolving of a small amount of gold at the margins of the particles.

TESTING FOR ACIDITY OR ALKALINITY

Acidity or alkalinity determinations of water for and solutions used in plants treating gold and silver ores are of importance. Methods range from simple tests with litmus papers or phenolphthalein solutions to pH determinations.

pH Control.—pH determinations are fairly simple and can be made easily and accurately by a plant operator. The pH scale may be likened to the Fahrenheit temperature scale on which freezing is 32 and values above and below represent degrees of heat and cold, respectively; on the pH scale, the dividing or neutral line between acidity (low) and alkalinity (high) is 7, the value of pure water. A pH value or hydrogen-ion concentration is simply a number denoting the degree of acidity or alkalinity of a solution. For example, assume that a solution has a pH of 7.6, which means that it is slightly alkaline. If another has a pH of 8.2, it is more alkaline than the one of pH 7.6. Therefore, an acid as hydrochloric, or an acid salt as alum, must be added to the solution of pH 8.2 to bring it to a pH of 7.6, and larger quantities must be added to bring it to the neutral point of 7.0 or to an acidity of pH 6.6 or 6.0. Similarly, if a solution has a pH of 6.0, it is acid, and it is necessary to add alkali, such as lime or soda ash, to bring it to the neutral point of pH 7.0. Larger quantities must be added to bring it to an alkalinity of pH 8.0 or 9.0. A solution that has a pH value of 5.0 is ten times as acid as one with a pH of 6.0, and one with a pH of 4.0 is ten times as acid as one of pH 5.0. Thus pH 4.0 indicates an acidity one hundred times as great as pH 6.0. A similar relationship holds on the alkaline side of the scale—which means that a solution of a pH 9.0 is ten times as alkaline as one with a pH of 8.0.

The principle of making pH measurements is based on the fact that various indicators change in color when they are acted upon by solutions of different acidities or碱性. Litmus is red in an acid solution (pH 4.6) and blue in an alkaline solution (pH 8.4). Similarly, chlorophenol red has a yellow color at pH 5.2 and a deep red color at pH 6.8; bromothymol blue is yellow at pH 6.0, greenish blue at pH 7.0 and deep blue at pH 7.6. Their color range is only 1.6 pH units, compared with 3.8 units for litmus, and therefore the former are much more distinct. Other indicators show pH values below and above the 5.2 and 7.6 values mentioned—down to 0.2 and up to 13.6. It is possible by means of bromothymol blue to determine whether a water or solution being tested is neutral, acid or alkaline. Then it is only necessary to proceed with another test using the more acid or alkaline indicator as the sample may require. In this manner the proper indicator for the solution is found.

Apparatus for the determination of pH is available from most chemical supply houses.

Elutriation Tests.—Because of the mechanical difficulties in making grading tests on sieves finer than 325 mesh, some precise method is desirable where finer grinding is being done, as at Kirkland Lake and Beattie (gold) and at Noranda (copper, gold, silver). Trials made with the Nobel elutriator at the latter place by C. G. McLachlan are described by him in *Trans. 112, A.I.M.E.*, 1934. The work was simplified by reason of sizing practically only one mineral—pyrite. At Noranda, they were able to establish five size zones of constant range below 325 mesh with this elutriator.

The Nobel elutriator (see Fig. 4) is a laboratory device for making particle-size classifications of mineral too finely ground to separate on any screen cloth. It accomplishes this classification by taking advantage of the settling rate of different-sized particles in vessels of varying capacity against a constant pressure head of water.

The apparatus, as described by McLachlan, consists of a set of four closed conical glass jars of increasing diameter, so connected that the overflow from the top of each jar forms the feed into the bottom of the next larger jar. The jars are supported in a water bath, in order to avoid temperature changes during the period of experimentation, since such changes affect the

viscosity of water and consequently the settling rate. The connections from the top of one jar to the bottom of the next larger one are made by copper tubes, at the high point of each of which is a riser tube of copper to allow the escape of entrapped air bubbles that might affect results and make them erratic.

A constant-pressure tank provides for a controlled flow of water through a series of jars and also supplies water at the same temperature to the water bath. The overflow from the

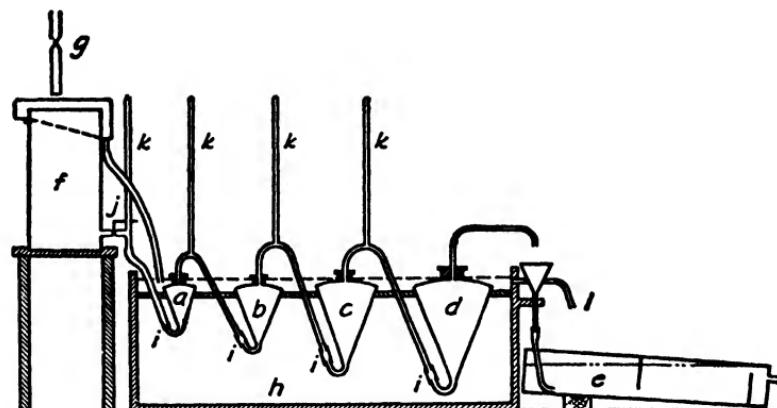


FIG. 4.—Sketch of Nobel elutriation apparatus; *a*, *b*, *c*, *d*—first, second, third and fourth elutriation jars; *e*, large pan for fifth elutriation product; *f*, constant-head tank; *g*, water supply; *h*, water bath; *i*, rubber connection; *j*, valve control; *k*, copper-pipe air vent from bends connecting jars; *l*, water-bath overflow to waste.

final jar is collected in a long pan, sufficiently large to allow an overflow of clear water if a flocculating agent, such as lime or zinc sulphate, is added to it.

In operation, a sample of 150 grams is placed in a small bottle containing 300 cc of water and $\frac{1}{2}$ gram of powdered sodium silicate. The bottle is agitated for a half hour, when the pulp is transferred to jar *a*, the water inlet of which is connected by a rubber tube to the outlet of the constant-head tank. The valve of this tank must be gradually opened to a rate of 150 cc per min., and this flow is maintained for 2 hr.; then the flow is increased to 250 cc for 22 hr. At the end of the 24-hr. period the content of each vessel is filtered, the residue is dried and weighed and the distribution of products is calculated. The method of particle sizing is either by making a count of the particles dispersed on a

slide or by direct measurement of reflected particles which have been imbedded and polished—particularly the latter method.

A typical size range for Nobel-elutriation series would be 4, 4½, 5½, and 6½ in. in diameter. Other types of elutriators are Schultze and Schoene which have cylindrical portions at the center and are therefore preferred by some.

TESTING FOR CYANIDATION

Examination of Ore.—Before an ore is tested for its amenability to the cyanide process it should be subjected to a preliminary examination in order that the experimenter may become familiar with its general physical and chemical characteristics. Knowledge so obtained will be of value in laying out the most effective testing program and may point to special methods of attack should the ore not respond readily to simple methods of cyaniding.

A representative sample is taken from the ore to be tested. The size of this sample will depend somewhat upon the amount of ore available, but it need not exceed $\frac{1}{2}$ to 1 lb. If the ore is in lumps coarser than 1 inch, examination may reveal the nature of the ore and gangue minerals, the degree of crystallization, the extent to which crushing may be required to liberate a portion of the ore minerals and other pertinent information. The sample is then ground to about 20 mesh, and a weighed amount panned. The concentrate from panning is roughly weighed and then examined under a low-power microscope. The amount and size of any elemental gold and silver are noted, and the nature and amount of any sulphides or heavy minerals are determined. Copper minerals and tellurides especially should be sought. A portion of the concentrate (or some of the crude ore) is given a rough qualitative chemical examination.

From such a procedure, information as to the desirability or necessity for any treatment such as amalgamation or concentration (gravity or flotation) preceding cyaniding may be obtained. Also, there will be some indication as to the degree of grinding necessary.

After being ground to a suitable size the ore is now sampled for assay. This size will depend upon several factors, such as the nature and amount of material available. It is essential that the assay sample be thoroughly representative, and the exact

procedure must be adapted to the conditions prevailing. The amount of analytical work done will depend upon the extent of the testing work to be undertaken and will range from determination of gold and silver only to a practically complete analysis.

In connection with the analytical work it is generally advisable to make a qualitative test for water-soluble salts and if any be present to determine their nature.

Sieve Analysis.—A sieve analysis of the ore will often supply information of considerable value. This may start at about 20 mesh and include the range of sizes to and including 325 mesh. A portion of each sieve size should be examined under the microscope and assayed for gold and silver.

A sample should also be separated into a sand portion and a slime portion. A series of such separations may be made, each starting with the ore crushed to a different degree of fineness. For example, a sample of ore might be crushed so that it all passes a 20-mesh sieve. The material is then separated by panning or by decanting under definite and controlled conditions into sand and slime fractions. Each fraction is weighed and assayed. Another sample of the ore might be ground to pass 65 mesh, and a similar separation made of sand and slime. Yet another separation might be made on the ore ground to pass 150 mesh. In this way the distribution of the precious metals can be determined under different degrees of grinding, and the possibilities for separate treatment of sand and slime thus indicated.

As a rule, preliminary cyanide tests are best made by agitation methods alone, and later the possibility of leaching a portion of the ore can be investigated.

AGITATION TESTS GENERAL METHODS

Procedure.—The following procedure has been found satisfactory for carrying out cyanide tests by agitation.

Winchester bottles of about 2.5- to 4-liter capacity are used. It is convenient to number the bottles and determine their tares, etching the figures on the bottle by means of hydrofluoric acid and then marking over the etching with a china- or glass-marking pencil. When a wet pulp such as the tailing from amalgamation or concentration tests is the material to be tested

by cyaniding, the weight of the pulp is obtained by weighing bottle and contents and subtracting the tare. When the test is finished, the solids are dried and weighed, and the amount of solution used may be determined. A weighed amount of ore is put into the bottle, and then a weighed amount of lime of known available lime content is added. Next is added a measured amount of a cyanide solution of known strength. The bottle with the pulp is then agitated.

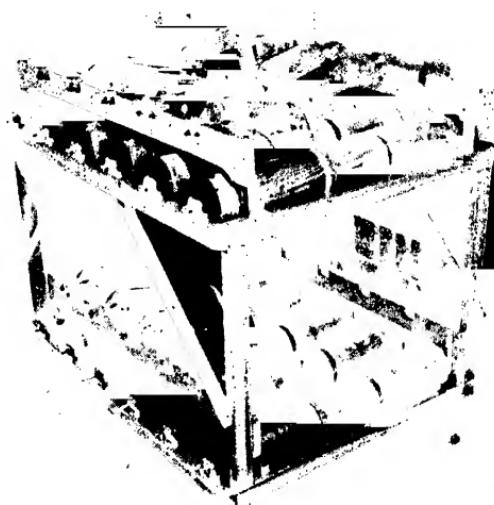


FIG. 5.—Shows a device used in the testing laboratory of The Dorr Company for making bottle cyanide agitation tests. This simple device can rotate as many as 40 bottles at one time.

The agitating device (Fig. 5) consists essentially of a series of horizontal wooden rollers about $4\frac{1}{2}$ in. in diameter and 30 in. in length, encased in old tire tubes and mounted on bearings with their axes parallel. They are spaced at about 7-in. centers and rotate at 50 to 75 r.p.m. When a bottle containing pulp is placed between two adjacent rollers it is caused to revolve, and the pulp is agitated. It is convenient to have a board set over the ends of the rolls and at a right angle to their axes with a notch midway between the centers of the rolls to engage the neck of the Winchester bottle. The notches are at such a height that the bottles are tilted, allowing them to be filled with more pulp than if they rested on their sides and with no danger of the pulp's splashing out.

Details of Testing.—Enough ore for the maximum probable number of tests is prepared by dry grinding so that it all passes a certain size, say 100 mesh. Samples for assay and screen analysis are taken.

We shall assume that it is desired to determine the extraction obtained by treating a certain ore with a solution containing 0.05 per cent sodium cyanide NaCN, and 0.025 per cent CaO (protective alkalinity), which means that the solution is to contain 1.0 lb. NaCN (free cyanide) and 0.5 lb. CaO (protective alkalinity) per ton of solution. The pulp is to be agitated at a dilution of 2.5 parts of solution to 1 part of ore for 24 hr.

A 400-gram sample of this ore is weighed out and put into a bottle. (Four hundred grams is a convenient amount, although any amount from 100 grams to 10 kilograms may be taken, bottles of suitable size being used.)

The next step is to add the lime. It will be assumed that preliminary tests or other information indicate that the ore will require about 2.0 lb. CaO (100 per cent basis) per ton for neutralization during the first hour of agitation. This means that 400 grams will require 0.4 gram CaO. In addition to the lime for neutralization, enough lime will be needed to bring the solution to 0.025 per cent CaO. At a dilution of 2.5 to 1 there will be 1000 cc (or ml) of solution which, at 0.025 per cent CaO, will require 0.25 gram. The total lime to be added then is $0.40 + 0.25$ or 0.65 gram. The hydrated lime has been tested and found to contain 69.0 per cent available CaO; therefore, we shall add $0.65/0.69 = 0.94$ gram of hydrated lime.

Water is added next, and the amount is determined as follows: 400 grams of ore requires 1000 cc of solution for a dilution of 2.5 to 1; 1000 cc of solution at 0.05 per cent NaCN contains 0.5 gram NaCN, equivalent to 50 cc of the stock 1.0 per cent NaCN solution. Therefore, add 950 cc water.

The pulp is shaken in the bottle for a moment, and then 50 cc of the 1.0 per cent cyanide solution is added.

Sometimes it is desirable to check the dilution, and this may be done readily by weighing bottle and contents if the tare of the bottle is known.

The bottle is now placed on the rollers and agitated, the time being noted. At the end of a period varying from $\frac{1}{2}$ to 2 hr., depending upon the rapidity with which the cyanide or the lime

is consumed, the solution is titrated for free cyanide and protective alkalinity.

This is done by removing the bottle from the rollers and allowing the pulp to settle until some of the supernatant solution can be drawn off in a pipette. If the solution is clear enough, a 10- or 25-cc sample may be *cautiously* drawn off in a pipette and titrated direct. If the solution is turbid, about 50 or 60 cc may be drawn off, using a 100-cc pipette. The solution is filtered through a dry filter into a clean, dry beaker, and a sample is taken for titration.

Determination of Free Cyanide.—Standard silver nitrate solution is made by dissolving 4.33 grams silver nitrate AgNO_3 , in distilled water and making up to 1 liter. The reaction between silver nitrate and sodium cyanide is represented by the following equation:



Thus, 169.9 grams AgNO_3 saturates 98 grams NaCN , or 4.33 grams AgNO_3 saturates 2.5 grams NaCN .

One cubic centimeter of the foregoing solution ($= 0.00433$ gram AgNO_3) saturates 0.0025 grain NaCN . Therefore, if 25 cc mill solution be taken for titration, each cubic centimeter of silver nitrate solution required will equal 0.0025 gram NaCN or 0.01 per cent NaCN . For example, a 25-cc sample of cyanide solution is titrated, and it is found that 4.8 cc silver nitrate solution is used, the strength of the cyanide solution then is 0.048 per cent NaCN . Sometimes it is more convenient to use a 10-cc sample for titration when strong solutions are used. In such cases the same silver nitrate solution is used, and the burette reading is multiplied by 2.5.

The solution to be tested should be clear (filter if necessary). Twenty-five cubic centimeters is put into a clean, transparent, 125-cc Erlenmeyer flask. The solution should not be diluted. The silver nitrate solution is added until the end of the reaction is indicated by the first appearance of a bluish haze, dulling the original brilliancy of the solution. This point is best determined against a black background, and the operator should experiment with various conditions of light until he is able to obtain consistent readings and check himself and others within at least 0.1 cc on the burette. A good setup is to place the burette

against a window with a good light (but not in the direct rays of the sun) so that the flask will be about level with the eye against a black background.

Determination of Protective Alkalinity.—Protective alkalinity is usually defined as the “alkaline hydrates and half the mono-carbonates.” The effect is to protect the cyanide from decomposition by acid constituents of the ore and by carbon dioxide in the atmosphere.

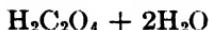
Either oxalic acid or a mineral acid may be used as a standard. If oxalic acid be used, the reaction is



Thus, 126 grams $\text{H}_2\text{C}_2\text{O}_4 \cdot 2\text{H}_2\text{O}$ saturates 56 grams CaO, or 5.62 grams $\text{H}_2\text{C}_2\text{O}_4 \cdot 2\text{H}_2\text{O}$ saturates 2.5 grams CaO. Therefore, if a solution containing 5.62 grams $\text{H}_2\text{C}_2\text{O}_4 \cdot 2\text{H}_2\text{O}$ per liter is made up, 1 cc of the solution (= 0.00562 gram oxalic acid) will saturate 0.0025 gram CaO. If 25 cc of mill solution be taken for titration, each cubic centimeter of acid required will equal 0.0025 gram CaO or 0.01 per cent CaO. For example, a 25-cc sample of solution is titrated, and it is found that 3.6 cc standard acid is used. The protective alkalinity of the solution then is equivalent to 0.036 per cent CaO.

An oxalic acid solution containing 5.62 grams $\text{H}_2\text{C}_2\text{O}_4 \cdot 2\text{H}_2\text{O}$ per liter has a normality of 0.0892. A mineral acid such as H_2SO_4 or HNO_3 of the same normality will serve equally well. The acid used should be standardized against a standard alkali.

A normal solution is one of which one liter contains a quantity of the substance, expressed in grams, equivalent to one gram hydrogen. When the solution is to be made of a salt that contains water of crystallization, the weight of such water must be taken into consideration. As in the case of oxalic acid:



The total atomic weight, including the water, is 126. Therefore, $\text{H}_2 = 126$, and $\text{H} = 63$ = grams of the salt to be added to one liter of water to make a normal solution. A decinormal solution contains one-tenth of this amount, or 6.3 grams per liter.

Phenolphthalein (1 gram phenolphthalein in 50 cc alcohol, dilute to 100 cc with water) is used as the indicator, as it gives

a value for monocarbonates corresponding with the definition of protective alkalinity already given.

The determination (Clennell's method) is made on the same portion of solution used for the free cyanide test as outlined. After the required amount of silver nitrate has been added for the determination of free cyanide, a drop or two of the phenolphthalein indicator is added. If the solution be alkaline, a pink color results. The standard acid is then run in until the pink color disappears, and the burette reading is noted.

Determination of Consumption of Cyanide and Lime.—The following notes of a typical test will illustrate the method of calculating the consumption of cyanide and lime:

Ore—400 grains.

Lime added—0.94 gram hydrated lime = 0.65 gram CaO.

Solution—1000 cc (950 cc H₂O, 50 cc 1.0 per cent NaCN solution).

Solution strength to be maintained at about 0.05 per cent NaCN and 0.025 per cent CaO.

9:00 A.M.—start

10:00 A.M.—25 cc taken out for titration, giving 0.042 per cent NaCN, 0.022 per cent CaO.

The solution is now brought back to a volume of 1000 cc having a strength of 0.05 per cent NaCN and 0.025 per cent CaO.

975 cc at 0.042 per cent NaCN = 0.41 gram NaCN; 1000 cc of solution should contain 0.50 gram NaCN; therefore, add 0.09 gram NaCN or 9 cc 1 per cent NaCN solution and 16 cc H₂O.

975 cc at 0.022 per cent CaO = 0.21 gram CaO; 1000 cc of solution should contain 0.25 gram; therefore, add 0.04 gram 100 per cent CaO or 0.04/0.69 = 0.058 gram hydrated lime.

1:00 P.M.—25 cc taken out for titration, giving 0.048 per cent NaCN and 0.02 per cent CaO.

The cyanide strength is close enough to that desired so that no correction is necessary. The protective alkalinity, however, is lower than desired and is raised as follows:

975 cc at 0.02 per cent CaO = 0.195 gram CaO. This amount subtracted from 0.250 gram (1000 cc at 0.025 per cent) = 0.055 gram; therefore, 0.055/0.69 or 0.08 gram hydrated lime is required. As the pulp may be expected to continue to consume lime for the next few hours, a slight excess over the theoretical amount may be added, say a total of 0.10 gram. Then, 25 cc of water is added to bring the volume of solution back to 1000 cc.

9:00 P.M.—25 cc taken out for titration, giving 0.045 per cent NaCN and 0.023 per cent CaO.

To bring the solution to the desired strength there are required

$$0.50 - 975 \times 0.00045 = 0.06 \text{ gram NaCN}$$

and $0.25 - 975 \times 0.00023 = 0.026$ gram CaO. As the pulp is now to be left to agitate overnight without further attention, add slightly more than these amounts to take care of consumption, say 0.08 gram NaCN or 8 cc 1.0 per cent solution and 0.05 gram hydrated lime. Also add 25 minus 8 or 17 cc water to bring the solution volume to 1000 cc.

9:00 A.M.—finish of 24-hr. agitation. The solution titrates 0.049 per cent NaCN and 0.023 per cent CaO.

Results may be calculated as follows:

NaCN In	NaCN Out
50.0 cc 1% solution	25 cc at 0.042% NaCN = 0.01050 gram NaCN
9.0 cc 1% solution	25 cc at 0.048% NaCN = 0.01200 gram NaCN
8.0 cc 1% solution	25 cc at 0.045% NaCN = 0.01125 gram NaCN
—	1000 cc at 0.049% NaCN = 0.49000 gram NaCN
67.0 cc 1% solution	—
or	—
0.67 gram NaCN	0.52375 gram NaCN

$0.67 - 0.524 = 0.146$ gram NaCN consumed by 400 grams ore. This is equivalent to $0.146 \times 5 = 0.73$ lb. 100 per cent NaCN per ton ore.

CaO In	CaO Out
0.940 gram hydrated lime	25 cc at 0.022% CaO = 0.00550 gram CaO
0.058 gram hydrated lime	25 cc at 0.020% CaO = 0.00500 gram CaO
0.100 gram hydrated lime	25 cc at 0.023% CaO = 0.00575 gram CaO
0.050 gram hydrated lime	1000 cc at 0.023% CaO = 0.23000 gram CaO

or $1.148 \times 0.69 = 0.792$ gram CaO
 $0.792 - 0.246 = 0.546$ = gram CaO consumed by 400 grams ore. This is equivalent to $0.546 \times 5 = 2.73$ lb. of 100 per cent CaO per ton of ore.

It may not be necessary or particularly informing to carry out tests to such a detailed degree as that shown. Each experimenter may determine for himself the extent to which it is desirable to correct solution strength and dilution during testing.

Rather than maintain the solution strength at approximately a predetermined figure, some experimenters prefer the more simple method of not making corrections of the solution during the period of agitation, starting with a solution high enough in cyanide and lime so that at the end of the period the solution is considered to be adequate. Such a procedure, however, is open to objection on the ground that the solution may be too strong at the beginning and too weak at the end, and these conditions may lead to erroneous deductions.

Series Testing.—After a preliminary test has indicated the consumption of cyanide and lime, several series of tests may be outlined in which the effect of changing one variable is determined. These variables may be: cyanide strength; protective alkalinity; fineness of grinding; time of agitation; dilution; addition agents (lead or mercury salts, bromocyanide and others).

Each series should embrace 4 to 6 deg. of the variable or as many as may be believed to be desirable. Finally, several tests may be made using various combinations of the best conditions as determined by the series tests.

Addition Agents.—With certain ores it may be found that the addition of a small amount of a lead or mercury compound is beneficial, either in purifying the solutions or in increasing the extraction or both. Lead in the form of litharge, as the nitrate or as the acetate, and mercury as the metal or oxide, may be used. Mercury is now seldom used.

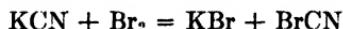
It is always worth while to make parallel tests with and without one of these agents. One-half to 1 gram of litharge or 4 or 5 grams of mercury added to 400 to 1000 grams of ore is ample. In practice, probably less than $\frac{1}{4}$ lb. of litharge per ton of ore is generally enough. As the effect of litharge may vary greatly with the amount used, it is of great importance to determine accurately the critical amount to use.

Sometimes it is desired to determine the effect of bromocyanide on certain gold ores, particularly those containing tellurides. As bromocyanide is an unstable compound, it must be freshly prepared before use; and as it and its vapors are extremely poisonous, due care should be used in its preparation and use. When bromocyanide is being used the protective alkalinity should be kept at the lowest possible point. In this connection a parallel test of the same low alkalinity but without the bromocyanide should be made to make sure whether possible improved results should be ascribed to bromocyanide or to low alkalinity. The reason for maintaining a low alkalinity when bromocyanide is used is because BrCN is rapidly decomposed by alkali. The reaction is usually illustrated thus:

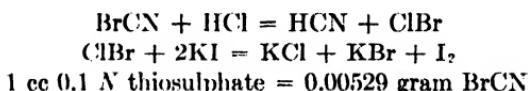


In his *Manual of Cyanidation*, E. M. Hamilton says this:

The usual method of making the reagent for laboratory use is to add a strong solution of cyanide to bromine (and not conversely) until the brown color is just discharged:



The quantity of BrCN may be determined in a working cyanide solution by acidifying with hydrochloric acid, adding an excess of potassium iodide and titrating the liberated iodine with decinormal sodium thiosulphate. J. E. Clegg in his *Chemistry of Cyanide Solutions*, 2d ed., gives the following reactions:



The BrCN is added to the cyanide solution to be used for extraction purposes in the proportion of about 1 BrCN by weight to 4 of KCN.

Change of Solution.—It is often of interest to determine the effect of a change of solution upon the extraction (rate and amount) and the chemical consumption. The procedure of effecting such a change will depend primarily upon the dilution of pulp during agitation and the dilution to which the pulp will settle after standing a short time. It is preferable to remove the solution by decantation rather than by filtration.

Precise manipulation is necessary in effecting a change of solution. Deductions as to the effect of this procedure should be made with caution, the practicability of reproducing the effect in plant operation being kept in mind.

Grinding in Solution.—The small samples available for laboratory experiments cannot be expected to reproduce exactly conditions that will exist in large-scale milling practice. Actual size reduction of the ore can be duplicated, of course, but closed-circuit grinding with a high circulating load would be very difficult to reproduce on a laboratory scale. Particularly is this true when grinding in cyanide solutions is to be practiced.

This is not to say that grinding in cyanide solutions should not be performed in the laboratory. On the contrary, such grinding is likely to add considerably to the knowledge accumulated in testing, and it should be done. But the data thus gathered should be weighed with mental reservations and

regarded as subject to some variation. As a matter of fact, laboratory grinding in solution is standard practice at some testing plants, and the information gathered is regarded as valuable.

Perhaps the nearest approach to solution grinding in the laboratory is to grind in water, classify to desired sizes and settle. Then cyanide solution and lime are added to make up a solution of the desired strength. This pulp may be then agitated for a time, $\frac{1}{2}$ to 1 hr., to check the approximate dissolution in the mill-classification circuit. After this the pulp may be thickened to the dilution at which the agitation and aeration are to take place.

Cyclic Use of Solution.—After preliminary batch tests have indicated the optimum conditions such as fineness of grind, dilution, strength of solution and time of agitation, tests should be made in which cyclic operations of agitation, thickening, decantation, washing and precipitation are carried out. The important point here is the reuse of the solution. It may be found that fouling of the solution takes place and its efficacy diminishes to a greater or lesser degree. If such be the case, it is essential that some method of reactivating or purifying the solution be determined to avoid the necessity of discarding too large an amount.

Preparatory Processes Followed by Cyanidation of One or More Products.—The most successful treatment process is the one that yields the greatest net profits. As the complexity of a treatment increases, so also does the number of factors to be considered before arriving at a decision as to the methods best adapted to the ore under examination.

The following operations may precede cyanidation:

1. Amalgamation. Amalgam converted to bullion; tailing cyanided.
2. Corduroy or blanket strakes. Concentrate amalgamated; tailing cyanided.
3. Gravity concentration. Concentrates may be
 - a. Smelted.
 - b. Amalgamated with or without cyanidation of tailing.
 - c. Cyanided, raw or roasted; tailing cyanided.
4. Roasting. Calcine cyanided direct or after amalgamation or concentration on blanket or corduroy strakes.
5. Acid wash to remove soluble harmful constituents such as ferrous iron and copper compounds.

6. Aeration with lime solution.
7. Flotation. Concentration treated as in 3; tailing cyanided or discarded.

Laboratory testing of the foregoing operations may be conducted as follows:

1. *Amalgamation*.—Several lots of the ore are prepared by grinding to such different sizes as may seem desirable. A 200-gram sample of ore is put into a bottle with 100 cc water, 1 gram NaOH and 30 to 50 grams mercury. The bottles are placed on rollers and revolved for 1 to 2 hr. The pulp is then panned, and the mercury separated. A small amount of sodium amalgam will be found useful in causing all of the mercury globules to coalesce. The tailings are dried and assayed. The mercury is dissolved in nitric acid, and any remaining gold washed, dried, and wrapped in lead foil with enough silver for parting, and then cupceled.

Sodium amalgam may be prepared by forcing small shavings of metallic sodium beneath the surface of mercury. Care should be taken to see that the mercury is dry. As the reaction is rather violent, due precautions should be taken in making the amalgam.

Amalgamation tests may also be made in a grinding pan or an iron mortar, particularly when the ore contains much silver. Copper sulphate and salt may be added to the pulp. The ore may first be given a chloridizing roast.

2. *Corduroy or Blanket Strakes*.—In testing the use of blankets or corduroy the pulp is wet ground to various sizes, and the pulp at a dilution of about 4 or 5 to 1 is passed over the strake set at an inclination of about $1\frac{1}{2}$ to $1\frac{3}{4}$ in. per ft. Whenever possible, the strake should be not less than 10 to 12 ft. long; the width is not so important. If for any reason it is not possible to use such a length, or if the amount of ore is limited, some idea of the effect of the strakes may be obtained by repeatedly passing the pulp over a short strake.

The tailings are collected, weighed, and assayed. The blankets are washed, and the concentrate collected. To make a complete recovery of the concentrate, the blanket after being carefully washed should be burned, and the ash added to the bulk of the concentrate.

3. *Gravity Concentration*.—If enough ore is available, a laboratory Wilfley table may be used. For small lots of ore a gold

pan is convenient and gives excellent results. Sufficient concentrate should be prepared so that the various methods of its treatment as already noted may be tried.

In handling concentrate the experimenter should be particularly careful to avoid losing any fine free gold.

Both concentrate and tailing may be tested with and without regrinding. The concentrate should be dried at low temperatures, weighed and assayed and calculation made as to the ratio of concentration.

4. *Roasting*.—In the laboratory, roasting experiments may be carried on by using fire-clay roasting dishes or heavy sheet-iron pans, preferably in an electrically heated muffle. The charges, period and temperature should be carefully noted. Modern methods of temperature control have made possible greatly improved results in this field.

If the ore contains much arsenic, the amount should be determined before and after roasting. If it is desired to reduce the arsenic to an extremely low amount, the calcine is reroasted with charcoal. The calcine may be passed over blanket or corduroy strakes and then cyanided after removing soluble acidic salts or neutralizing them with lime.

5. *Washes*.—Some ores contain soluble iron or copper compounds which cause a high consumption of cyanide. A wash of dilute sulphuric acid or sulphurous acid may remove these compounds to permit of economical treatment of the ore by cyanidation. While acid pretreatment has been used in a few cases, most careful investigation should be made before adopting it.

6. *Aeration*.—On some refractory ores and concentrates a preliminary intense aeration in a strong lime solution before the addition of cyanide has been found beneficial. The maximum solubility of lime, CaO, in the ordinary cyanide mill solution is about 2.5 lb. per ton of solution. However, as much as 50 to 100 lb. of lime per ton of ore may be used, depending upon dilution and chemical consumption.

7. *Flotation*.—Flotation testing of ores containing precious metals may be either for the purpose of determining the applicability of the process to an ore under investigation or to determine the possibilities of improving the work of an operating flotation plant.

In the latter case the work is best done at the mill itself, samples taken directly from various points of the mill stream being used for comparison. In this way it is possible to obtain a pulp that has been ground under operating conditions as well as to use the mill water, which may be of great importance.

When the tests are made to evaluate the application of the process to an ore, the work may of course be done anywhere. However, it should be made certain that the ore used for experimentation is thoroughly representative of that which is to be milled. Furthermore, if possible, a sample of the water to be used in the mill should be obtained, and its effects noted.

One point that should always be kept in mind is the determination of the proper place flotation may fill in securing the maximum net profit from the ore. On some ores an all-flotation treatment might be indicated; usually, however, better results may be obtained by combining flotation—if it be used at all—with other methods such as amalgamation, concentration or cyanidation. Again, if all flotation seems to be best, the possibility of separate treatments of different fractions of the pulp should be considered. For example, the so-called primary slime might advantageously be separated and floated in a separate circuit.

Naturally, such points cannot be determined in advance; but if the operator will keep them in mind as the work progresses, he will be enabled to obtain a better perspective of the basic problem, which is to produce the maximum profit from the operation as a whole, and determine the proper balance between extraction and cost.

The first testing should be confined to an all-flotation method so that the experimenter may become familiar with the ore. Careful observation, coupled with experience in testing, will enable the operator to interpret the various phenomena.

It is desirable that he shall have had practical operating experience so that he may know the limitations of each step under consideration. Fortunately, it is generally true that if the laboratory procedure follows sound operating practice, the flotation-mill results will equal or surpass those attained in the laboratory.

The simplest flotation testing results when a single mineral or concentrate is desired. This is usually the case in copper

ores when the copper occurs as one or more of the sulphide minerals. Such ores generally contain more or less pyrite. If the pyrite be present in such an amount that it would affect the grade of the concentrate, then differential methods of flotation are employed whereby the copper metals are caused to float and the pyrite is prevented from floating.

Differential methods are also used in separating two or three minerals such as galena and sphalerite or galena, sphalerite and pyrite. When making differential separations it is important to know the degree with which one mineral may be intergrown or locked with another. This is determined by microscopic examination and chemical analysis.

The distribution of the precious metals or their association with different base metals should be determined, as this has an important bearing upon the net value of the concentrate produced. For example, gold or silver associated with a lead concentrate yields a greater net return than an equivalent amount in a zinc concentrate.

Within limits, the type of laboratory cell used is not a matter of great importance so far as obtaining indicative results is concerned. That this is so is shown by the fact that different operators using different types of laboratory machines with which each has become familiar will arrive at practically the same procedure. The important thing is to obtain a machine which is mechanically reliable, easy to clean and which permits quick work.

The amount of material taken for a test will be governed by the amount of ore available, size of machine, amount of floatable material in the ore and various other factors. If the amount taken for a test be too small, any errors or effects of manipulation are unduly magnified. On the other hand, if the amount be too large, each test may consume too much time—not so much the actual flotation itself but the grinding or other preparation of the pulp and the drying and sampling of the products. Probably 500 to 1000 grams is the best all-round size of charge. If possible, however, several different sizes of machines should be available so that charges of 50 to 2500 grams may be used if desirable.

Where differential separations are made or several cleaning operations seem called for, the amount of pulp taken should be

sufficiently large to allow the various operations to be made and to ensure resultant products large enough for assay.

Broadly speaking, the reagents used in flotation may be divided into three classes. Various writers on flotation extend the division, and the nomenclature differs somewhat. However, the three following classes are generally recognized and accepted by flotation operators without confusion:

1. Conditioning agents. These include such compounds as lime, soda ash, sodium silicate, sodium sulphide, sodium cyanide, sodium sulphate, sodium bichromate, copper sulphate and zinc sulphate, which are in general use. There are others less frequently used.

2. Promoters or collectors. Under this heading are found the various xanthates, dithiophosphates (aerofloat), thiocarbanilid and others classed as chemical promoters and also certain oils and tars.

3. Frothers. This class embraces such compounds as the various pine oils, cresols, alcohols and other petroleum and coal- or wood-tar derivations.

It is not within the scope of this chapter to enter into a detailed discussion of the effects and use of the many reagents used in flotation. It is assumed that either the experimenter is sufficiently familiar with the characteristics of the various reagents to make his own selection or else he knows little or nothing about them. In the latter case he can learn more by observation than in any other way. To such an one the suggestion is made that he obtain from the manufacturers or distributors of reagents data concerning their use. Then he should consult various books and publications in which details of flotation-plant practice are described.

The U. S. Bureau of Mines, in cooperation with various mining companies, has issued a number of technical papers covering in great detail the flotation operations at most of the important milling plants in the United States. Publications of the various mining and metallurgical societies and technical journals contain many valuable articles. Manufacturers' catalogues also are well worth study.

The amount of reagents used may be determined either by weight (dry reagent) or by volume (definite-strength solution). Lime is prepared as described under the sections on cyanide

testing. Soda ash may be used either dry or in solution. If dry, it should be thoroughly dried to constant weight and kept in a stoppered bottle. The moisture content should be checked occasionally, as soda ash may absorb 15 to 18 per cent water and still appear quite dry.

Water-soluble salts such as zinc and copper sulphates and sodium cyanide should be used as solutions of definite strength—1, 5 or 10 per cent—and measured by pipettes. Xanthates are best used as 1 per cent solutions and freshly prepared each day as needed.

Oils are measured as drops from calibrated pipettes or medicine droppers. A dropper is used for each oil and is calibrated by weighing 100 drops. It is convenient with some oils—particularly steam-distilled pipe oil—to have several droppers delivering different-size drops. This can be done by drawing out the tip of the dropper to a capillary and then breaking it off to obtain a small orifice. In this way it is possible to obtain a variation of 50 to 250 drops of pine oil per gram.

The determination of the pH value of the water of the pulp should always be made for reference, and the effects of changing the pH noted. A good procedure is to take a sample of the dry crushed ore and grind it with distilled water in a porcelain jar mill with flint pebbles. The pulp is filtered, and the pH value of the filtrate determined. At the same time the water is examined in more or less detail for soluble salts.

There are many pH sets available, varying from simple color-chart comparators to elaborate potentiometers, and each experimenter may determine for himself which best serves his purpose.

In order to obtain an idea of the grade of concentrate as well as the recovery that may be obtainable in practice, the following tests may be made:

Consider the simplest case where a one-mineral concentrate is desired. After a few preliminary tests so that the operator has a fairly comprehensive idea of such factors as degree of grinding, pH, reagents and other pertinent data, two tests are made. In the first one such conditioning reagents as may be required are added, and then three or four successive froths are removed and kept separate, staged additions of promoters and frothers being used. The several froths and the tailing are dried, weighed and assayed, and the distribution of the valuable mineral determined.

All products should be examined under the microscope to obtain an idea of the nature and amount of any locked or true middling grains.

A comparative test is then made under the same conditions, but only one froth is made, the same kinds and amounts of reagents being used as before. After the tailing has been discharged from the cell and the cell cleaned, the froth is returned and refloated. Further addition of reagents to this cleaning operation may or may not be made; only the judgment and experience of the operator can determine. This operation of cleaning the froth may be repeated as often as deemed necessary. All products are finally dried, weighed, assayed and examined as before.

The information so obtained may then be studied for the purpose of arriving at an idea of a possible flow-sheet, and the advantages and disadvantages of the various types of circuits may be determined. However, all such deductions should be made with caution and regarded only as indicative of what to expect in plant practice.

The most important phase of the entire testing procedure is the examination of the several products made, and this phase should receive the attention commensurate with its importance. The first thing to do is to make as complete chemical analysis of each product as the limitations of time and expense and the requirements of the information sought will permit. Probably next in value is a screen analysis of each product with chemical analysis of each screen size. Finally, the various screen products should be carefully examined under a microscope.

As flotation tests progress, the operator will obtain an idea of the nature of the results being obtained by visual examination both of the froth and of the tailing. This is best done by vanning samples on a white enameled vanning plaque. The plaque is then placed under a microscope, and the product examined.

The froth sample is readily obtained by scraping it on to the plaque. The tailing or pulp sample may be drawn from the cell by means of a glass tube about $\frac{1}{4}$ in. in diameter and 12 to 15 in. long, the ends of which have been closed sufficiently to hold the pulp after the operator has sucked it into the tube.

Complete notes on every test are essential. They are best kept in a bound notebook (an 8- by 11-in. ruled "composition"

book is excellent) with pencil (ink is almost sure to blot, and the notes become illegible, as the experimenter's hands will probably be wet). The first thing to enter is the date of each test. Full details are noted covering all pertinent factors such as method of grinding, size of grind, dilution of pulp, temperature, pH, kind and amount of reagents used and time of frothing.

The operator should also record in detail observations such as the character of the froth, the effect of each addition of reagents and other phenomena. Nothing is too unimportant to be noted, so that months after a test has been made the original experimenter or another may repeat the test and obtain substantially the same results; by reading the notes he may obtain an accurate and comprehensive picture of just what happened.

After the tests have been completed and a formal report is written, all pertinent data may be abstracted and assembled in a form designed to convey readily and accurately to the reader the necessary information.

With some ores the removal of the slime portion before flotation is extremely beneficial. Therefore, should an ore be encountered that seems to be difficult to float by usual methods, the effect of removing the slimy portion ahead of flotation should be investigated. Practically, slime may be considered as that portion of the pulp finer than about 50 microns.

In practice the slime is usually removed by means of a Dorr bowl classifier or a Dorr hydrosePARATOR. Sometimes a "mud" or a "talc" froth may first be removed by flotation. Such froth will contain the greater part of the deleterious components with a negligible amount of valuable mineral. The tailing from this operation will then generally respond to usual methods of flotation.

The removal of slime from a pulp often markedly improves differential separations, although the results may not economically justify the added step. Caution should be used in recommending this procedure.

After the laboratory work has revealed methods by which it is believed that the ore may be successfully treated by flotation, the work should be reviewed in the light of the relationship of flotation to other phases of the entire milling process.

A number of such other phases might be involved. For example, there are grinding, classifying, concentration (tables

or blankets and other gold-saving devices), thickening and filtering. In the case of ores containing gold or silver it may be found that a combination of flotation and cyaniding may yield a greater net return than either one alone. The effect of the various reagents on classifying, thickening and filtering should be investigated, particularly if it is desired to recover the water for reuse.

The equipment of a flotation laboratory may range from extremely simple to quite elaborate. The minimum requirements include some means of wet grinding and a flotation cell. More complete equipment may include various types of dry-grinding machines, wet-grinding units employing steel rods, steel balls and flint pebbles, several types and sizes of flotation cells, testing sieves, microscope, air compressor and vacuum pump.

The arrangement of the laboratory should be carefully considered so that there will be no lost motion in conducting a test. If possible, all wet work should be done on a table or bench covered with sheet zinc and sloping to a drain, so that any spills may be cleaned up readily.

Consumption of Flotation Reagents.—The following information concerning reagents and quantities commonly used in the flotation of gold ores of no particular refractoriness is furnished by the American Cyanamid Company.

	Pounds per Ton
Addition agents:	
Soda ash.....	0.50 to 3.0
Sodium silicate.....	0.50 to 2.0
Activating agents:	
Copper sulphate.....	0.10 to 1.5
Promoters:	
Aerofloat 15.....	0.05 to 0.15
Aerofloat 25.....	0.05 to 0.15
Aerofloat 31.....	0.05 to 0.15
Reagent 208.....	0.05 to 0.15
Reagent 301.....	0.05 to 0.15
Potassium amyl xanthate.....	0.05 to 0.15
Thiocarbanilid.....	0.05 to 0.15
Frothers and froth stabilizers:	
Pine oil.....	0.025 to 0.10
Cresylic acid.....	0.05 to 0.15
Coal-tar creosotes.....	0.10 to 0.25
Pine-tar oil.....	0.05 to 0.15

Filter Tests and Calculation.—The following table illustrates the technique of laboratory filter tests and the deductions gained therefrom. The conditions were: An ore pulp of 42 per cent moisture content, filtered on a leaf of 0.5 sq. ft. capacity at room temperature. The objective was a reasonably dry cake with maximum capacity and a filter to handle 100 tons of dry solids per day.

RESULTS OF LABORATORY TESTS

Conditions	Test			
	1	2	3	4
Cloth used.....	Twill 26			
Feed volume, cubic centimeters.....	1000	650	500	400
Cake forming, mixtures.....	2 $\frac{1}{4}$	1	2 $\frac{2}{3}$	1 $\frac{1}{2}$
Vacuum, inches.....	25	25	25	25
Cake drying, minutes.....	2 $\frac{1}{4}$	1	2 $\frac{2}{3}$	1 $\frac{1}{2}$
Vacuum, inches.....	18	20	23	25
Cake quality.....	Cracked	Slightly cracked	Slightly cracked	Good
Cake thickness, inches.....	1 $\frac{1}{2}$	3 $\frac{3}{8}$	1 $\frac{1}{4}$	1 $\frac{1}{2}$
Cake removal.....	Satisfactory			
Weight of wet cake, grams.....	1115	725	560	458
Moisture content, per cent.....	21	22	23	23
Weight of dry cake, grams.....	880	566	423	357
Total cycle, minutes.....	6 $\frac{3}{4}$	3	2	1 $\frac{1}{2}$
Capacity, pounds per square foot per 24 hr.....	828	1200	1370	1520

The results of tests 2 and 3 were selected and, a 40 per cent safety factor being allowed,¹ a filter area of 220 sq. ft. for 100 tons per day was recommended.

Settling-test Procedure.—The principles of settling-test procedure and the application of formulas to determine sizes of continuous thickeners are relatively simple and are briefly described below. However, detailed manipulations under specific problems and the application of test results to successful milling practice call for experienced interpretation.

¹ The factor of safety to apply to any individual test is a matter of judgment based on experience and will vary considerably.

The following formula expresses the relationship between the settling rates of pulp at various dilutions, in terms of thickener area required:

$$A = \frac{1.333 (F - D)}{R \times \text{sp. gr.}}$$

where A = square feet per ton of dry solids per 24 hr.

R = settling rate in feet per hour of a feed with F dilution.

sp. gr. = specific gravity of liquid.

F = weight ratio of liquid to solids for the rate R .

D = weight ratio of liquids to solids in discharge.

The zone requiring the greatest unit area is found by applying this formula to pulp of different densities, ranging in dilution from feed to discharge density. It is this zone that determines the area that must be provided for the pulp being tested.

Thickening Capacity.—The following formula is used to determine the volume provided in a tank in the thickening zone. Such volume depends directly upon the period of detention required for the sludge to reach the desired density:

$$V = \frac{4T(G - \text{sp.gr.})}{3G(S - \text{sp. gr.})}$$

where V = volume in cubic feet required for thickening per ton of solids per 24 hr.

S = average specific gravity of thickened pulp during compression period.

Sp. gr. = average specific gravity of solution.

G = average specific gravity of solids in pulp.

T = period of detention in hours.

Air Sizing.—Air sizing down to 10 microns, recently developed by Prof. H. E. T. Haultain of Toronto University, has been established as regular procedure in the testing laboratories of The Lake Shore Gold Mines, Ltd., at Kirkland Lake, Ont., with most satisfactory results. For details see the publication of the Canadian Institute of Mining and Metallurgy, 1936, "Milling Investigations at Lake Shore."

CHAPTER III

COARSE CRUSHING

Methods for the coarse crushing of ore range from a simple combination of a grizzly and one crusher to an elaborate arrangement in series of grizzly, jaw crusher, screens and rolls or cone crushers.

Crushing is done underground at a few large mines. In general, the jaw crusher is the most suitable primary machine, yet many gyratories do the first breaking. There is a wide choice of these machines. Wearing parts are of special steel, especially hard faced at the point of greatest abrasion. For secondary and tertiary crushing, gyratories and concushoppers are used, the latter particularly for final reduction but this can also be done by rolls which give a uniform product.

Interposed in a large crushing plant should be grizzlies, screens, feeders for crushers, magnets for picking up loose iron and steel and devices for weighing the ore.

An efficient crushing plant has grizzlies, trommels or shaking or vibrating screens to separate the ore that is fine enough to by-pass the next step. The grizzly—stationary, rotary or reciprocating—is suitable for the first stage, but thereafter the trommel or screen is more positive in its separation. Several flow-sheets are shown (Figs. 11 to 17) of different combinations of various stages in crushing. Few trommels are used, except those on dredges and in sand and gravel plants. The Hollinger has two sets of trommels, all with 1-in. round holes in manganese-cast plate. These are boxed in, and the dust is drawn off. It costs 0.6 ct. per ton to operate them. Openings in screens range from $1\frac{1}{2}$ in. diameter to 4 mesh or less. They may be of the simple shaking type or of the electric vibrating type. The latter are better for separating the finer sizes of ore.

On the Rand, mechanical grizzlies and vibrating screens have provided a better division of the mine-run feed for separate sorting of the coarse and fine waste and for two- and three-stage crushing. The closed circuit is being considered and is employed in one plant. More work is being done by crushers, and the final product is much finer.

SELECTION OF PRIMARY CRUSHERS

As primary crushing machines the choice is limited to two types—the jaw crusher and the gyratory. There are two types of jaw crusher, the Blake and the Dodge. In the former the moving jaw is swung at the top; in the latter, at the bottom. The Blake type, having a receiving opening of fixed size and a varying size of discharge, has greater capacity. The Dodge-type crusher, having a variable receiving opening and a fixed discharge, provides a more uniformly sized product. The Dodge crusher, owing to its small capacity, has practically disappeared from ordinary commercial-plant use, so that references to jaw crushers ordinarily mean only the Blake-type device. Dodge crushers are sometimes found in laboratories and assay offices but seldom elsewhere.

The Blake-type jaw crusher is, of course, a reciprocating-action machine, crushing rock by direct pressure between a fixed plate and a swinging jaw.

The gyratory crusher consists essentially of a circular shell with inner sides inclining toward a central orifice. A central shaft passes vertically through this opening and is hung centrally from a spider which spans the opening at the top. This shaft is eccentrically moved at the bottom and is equipped with a conical crushing head which operates between the inclined sides of the shell, crushing rock between the head and shell. The crushing operation in this machine is continuous and rapid. It is able to clear itself when completely buried in its feed.

In selecting the proper type of primary crusher a proper balance must be established on the basis of the largest size of rock to be crushed and the total quantity of material to be handled in a unit of time. It is also to be borne in mind that the product of the gyratory is much more uniform in size than that of the Blake-type jaw crusher.

Cone crushers are a modification of the gyratory type.

POSITION OF CRUSHERS

Coarse crushing or breaking is the primary mechanical step in ore reduction. It is occasionally done underground in large plants such as the McIntyre and Homestake mines. Where sorting is done it is usually at the shaft mouth, although central plants may be used.

On the Rand the older plants operate jaw crushers for the total crushing, but in the newer ones, jaw crushers break the ore prior to sorting, and gyratories are used for the finer reduction. Cone crushers are used in a few plants, and probably more will be installed.

While jaw and gyratory crushers compete for primary crushing, gyratory and cone crushers share the secondary field.

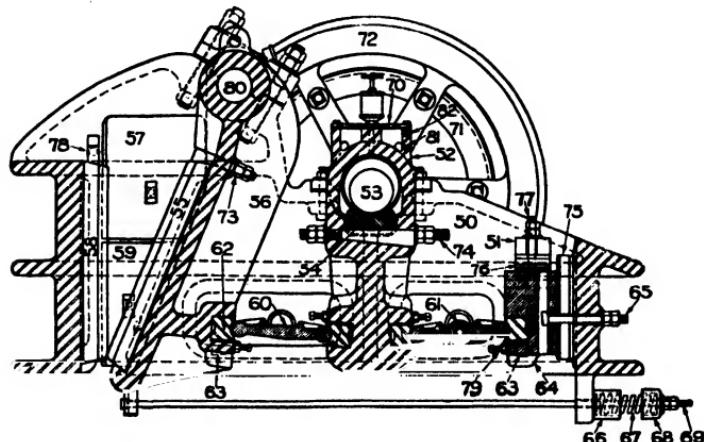


FIG. 6.—Principal parts of a Buchanan type "B" jaw-crusher.

50. Frame (steel)	68. Spring cap
51. Frame lugs for shims	69. Spring rod
52. Pitman (steel)	70. Water-jacket cover-plate (pitman)
53. Eccentric shaft	71. Pulley
54. Half box (pitman bearing)	72. Flywheel
55. Swing jaw-plate	73. Swing-jaw key-bolt
56. Swing jaw (steel)	74. Key-bolt (half box)
57. Upper cheek-plate	75. Rear shims, vertical
58. Stationary jaw-plate	76. Shims, horizontal (top)
59. Lower cheek-plate	77. Lug bolt for tail piece
60. Front toggle	78. Stationary jaw key-bolt
61. Rear toggle (safety)	79. Set-screw toggle wedge
62. Toggle bearings	80. Swing jaw-shaft
63. Toggle wedge-block	81. Water connections—inlet
64. Tail piece	82. Water connections—outlet (and drain)
65. Tail piece bolt	
66. Spring box	
67. Jaw spring	

TRAMP STEEL

Magnets are used for the collection of tramp steel mainly at large mills. At small plants most of the large pieces of steel are picked out of the ore by crusher attendants. At the McIntyre Porcupine, for example, a large magnet at the Symons crusher is fed by a 250-volt, direct-current generator at 24 amp. equivalent to $9\frac{1}{2}$ kw. A heterogeneous collection of iron and steel results.

WEIGHING ORE

Weighing of ore by automatic machines is likewise the practice of large mills. At small plants an occasional car of ore is weighed, and an average factor applied to all. Devices for weighing large quantities of ore at a constant rate have always

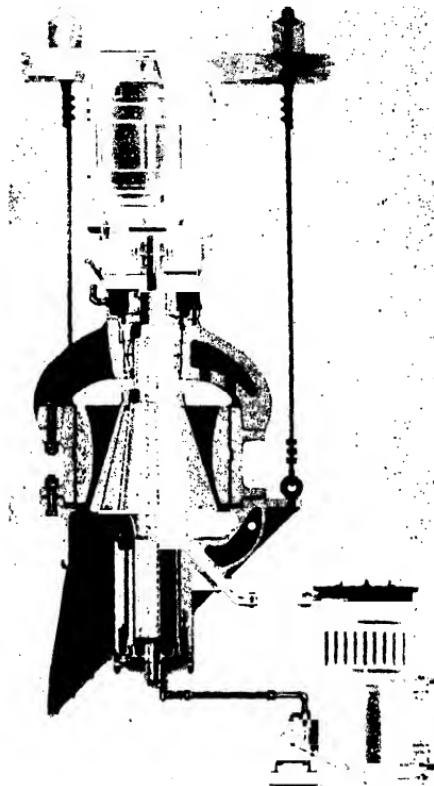


FIG. 7.—Type "B" Newhouse crusher, manufactured by Allis-Chalmers Manufacturing Company.

been considered expensive, but such quantities can be weighed with low error. Among apparatus for this purpose are the Blake-Dennison, Hardinge constant-weight and Merrick weightometer.

FEEDING CRUSHERS

There are roughly four periods in the normal cycle of coarse crushing not employing feeders: (1) The breaker receives a large

charge of ore; (2) operates under choke feed with the power at a maximum; (3) gradually discharges the crushed ore and (4) runs empty. This is not the most economical method yet is the practice in many crushing stations. Some method of feeding a crusher regularly is preferable. This may be done by means of a feeder or gate to the bin above the machine or by a heavy chain, a pan feeder, a shaking grizzly or an apron feeder, which occupy little head space. The objective is to insure a regular feed to

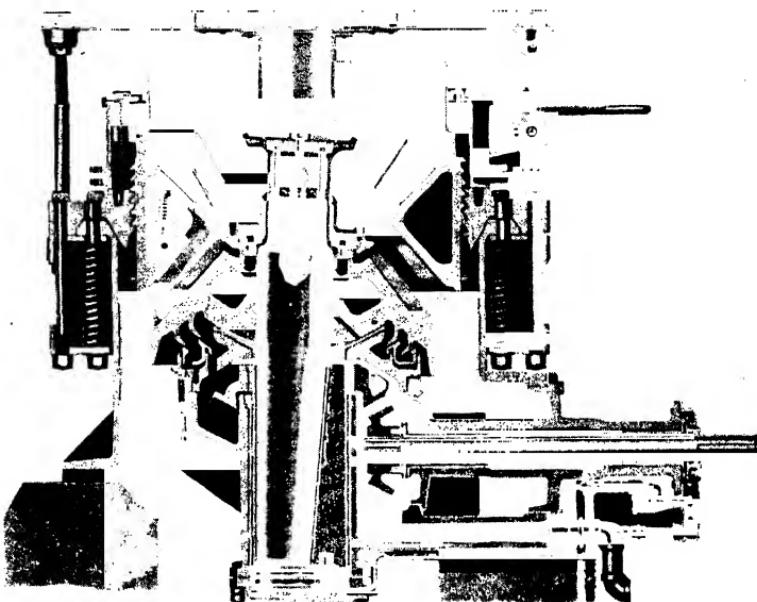


FIG. 8.—Sectional view of short-head Symons crusher.

keep the crusher steadily occupied and thus flatten the power curve. Figure 9 shows an apron feeder-jaw crusher arrangement. Cone crushers and rolls, particularly, may be fed by belts; cone crushers sometimes are fed from chutes.

Ore for the underground crusher in the McIntyre-Porcupine mine is fed by a Ross chain feeder consisting of six lengths weighing $5\frac{1}{2}$ tons. The feeder is driven by a 5-hp. motor through a bronze worm reducer, with a sprocket chain-gear and pinion arrangement. This gives a speed of 19 r.p.m. and a capacity of 175 tons an hour.

At Wiluna, Western Australia, Ross chain feeders deliver 10-in. ore to the primary crushers at the rate of 130 tons per hour and 4-in. ore to the secondary crusher at the rate of 40 tons.

The primary feeder consists of five chains 28.3 ft. long, running at $7\frac{1}{2}$ ft. per min. and using 4 hp.; the secondary feeder has five chains 14.6 ft. long, running at 8 ft. per min. and using 2 hp. The cost of feeding is 0.037 pence or 0.074 ct. per ton.

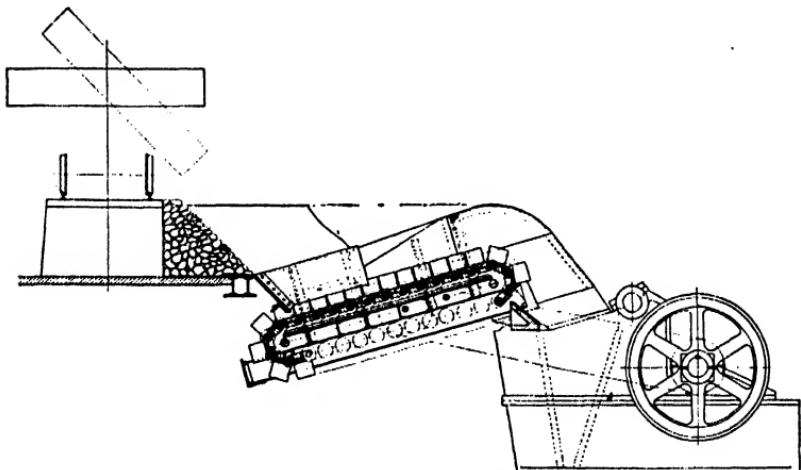


FIG. 9.—Apron-feed conveyor for Superior jaw crusher.

CRUSHING PRACTICE

FIRST STAGE

Crushing Underground.—1. The Homestake is a good example of this practice. At the 800-, 1400- and 2000-ft. levels of the Ellison shaft are 36- by 48-in. Traylor jaw crushers, set to $3\frac{1}{2}$ in., which are fed with plus 6-in. ore from grizzlies. The discharge mixes with the grizzly undersize and is hoisted to the surface. These crushers are driven by 125-hp. motors and break 280 tons per hour.

Crushing on Surface.—1. First-stage crushing at Loreto, Pachuca, Mexico, of an ore containing more than 70 per cent silica, consists of passing the run-of-mine ore, maximum size 12 in., over a grizzly with 4-in. openings. A 20-in. Traylor gyratory crusher reduces the oversize to 4 in. Although driven by a 100-hp. motor, only $28\frac{1}{2}$ hp. is used.

2. At the Golden Cycle custom plant, Colorado Springs, treating Cripple Creek and other ores, a $5\frac{1}{2}$ -ft. Symons cone crusher is fed with ore of $3\frac{1}{2}$ to 4 in. maximum size which has passed over a stationary screen having $\frac{5}{8}$ -in. holes. The Symons produces 100 tons an hour of $\frac{1}{2}$ -in. material for further comminution.

3. At the Premier mill, British Columbia, the mine-run ore is dumped on a grizzly of 80-lb. rails, spaced at 12 in. A Stephens-Adamson apron feeder delivers the grizzly oversize to an 18- by 30-in. jaw crusher set at 3 in.

4. At the Ross shaft of the Homestake mine, the primary crushing is done by two Allis-Chalmers size-8 gyratories. They are driven by 70-hp. motors and reduce the ore to $4\frac{1}{2}$ in.

5. At Morro Velho, Brazil, the primary jaw crusher, with 18- by 27-in. opening, receives plus 5-in. ore from a bar grizzly and crushes it for secondary reduction.

6. The two surface-crushing plants of the Teck-Hughes at Kirkland Lake receive ore from an underground station. Jaw crushers, grizzlies, vibrating screens and Symons cone crushers produce 1200 tons or more daily of $3\frac{1}{4}$ -in. ore for the rod mills.

7. The primary crusher at the Presidio mine, Texas, is a belt-driven Telsmith 8-A gyratory. It is fed with ore not larger than 7 in. and is set at $1\frac{3}{4}$ to $2\frac{1}{4}$ in. The hourly capacity is 35 tons.

SECOND STAGE

As large rock breakers will not reduce large pieces of ore economically to a size suitable for feeding to a tertiary crushing machine, some intermediate machine is necessary. This is termed secondary crushing and may be done by jaw or gyratory breakers or by cone crushers.

1. At the Presidio mine, Texas, a Symons cone crusher, set at $\frac{1}{4}$ in., reduces $1\frac{3}{4}$ - to $2\frac{1}{4}$ -in. ore from a gyratory breaker, at the rate of 25 tons per hr., and uses 32 hp.

2. In the surface plant at the McIntyre-Porcupine a 7-ft. Symons cone crusher (see Fig. 8), set at $\frac{7}{16}$ in., crushes material from the mine breaker, of which 16 per cent is plus 6 in., 8 per cent plus 4 in., $17\frac{1}{2}$ per cent plus 2 in., 6 per cent plus $1\frac{1}{2}$ in. and $3\frac{1}{2}$ per cent plus 1 in. Of the discharge, about 70 per cent remains on sieves down to 8 mesh. A 200-hp. motor with V-belt connection drives the crusher.

3. Second-stage crushing at Loreto, Pachuca, Mex., is as follows: 4-in. ore from the first crusher, combined with the undersize of a grizzly above it, is passed over a 2-in. grizzly the oversize of which is fed to secondary crushers and the undersize to vibrating screens with $1\frac{1}{4}$ -in. clear openings. Two $5\frac{1}{2}$ -ft. Symons cone crushers, set to give a $\frac{7}{8}$ -in. product, do the secondary breaking. Each is driven by a 225-hp. motor and V belt and consumes 172 hp. Mantles last 85 days, and bowl liners 87 days.

4. At the Premier mine, British Columbia, 3-in. ore from the primary breaker passes over a Niagara screen with 1-in. openings. The oversize is fed direct to a size-6 McCully crusher set at 1 in.

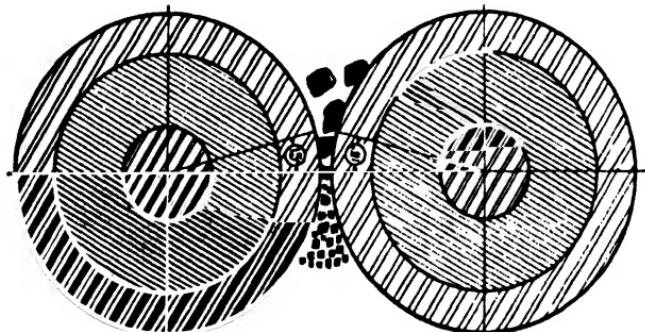


FIG. 10.—Crushing action of rolls, showing 4 to 1 reduction.

5. At the Ross shaft of the Homestake mine, the $4\frac{1}{2}$ -in. ore from the primary breakers passes over a grizzly to two 7-ft. Symons cone crushers, set at $1\frac{1}{2}$ in.

6. At Morro Velho, Brazil, 5-in. ore from the primary jaw crusher and undersize from a grizzly is reduced to 2 in. in three 8- by 15-in. breakers of similar type and then is fed to the stamps. Coarse crushing (including haulage) costs 4.26 pence (8.52 cts.) per ton.

THIRD STAGE

The third step in coarse crushing, or tertiary reduction, depends upon what is required. Two-inch or smaller pieces from the secondary crushers may be fed direct to stamps, rolls,

cone crushers, rod mills or ball mills, but the trend appears to be to use a third coarse crusher, to reduce the ore to $\frac{1}{4}$ - or $\frac{1}{2}$ -in.

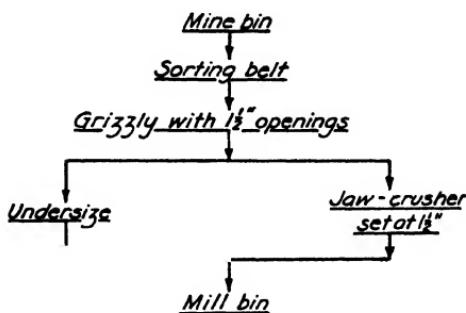


FIG. 11.—A simple crushing plant for 90 tons per day.

size, particularly for ball milling. For such work the Symons machine is well suited, just as it is for coarser reduction.

At the McIntyre Porcupine, according to P. D. P. Hamilton, *Trans. 112, A.I.M.E.*, the ore is first crushed to 7-in. size under-

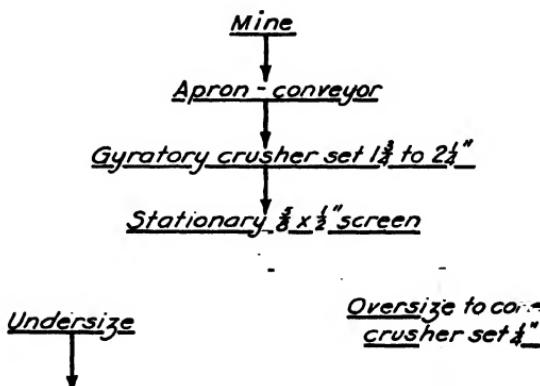


FIG. 12.—Two-stage crushing by gyratory and cone crushers, with screen in open circuit.

ground. Thence it passes through a cone crusher, screens and rolls, which give the following sizes:

SIEVE ANALYSIS OF CRUSHER-HOUSE PRODUCTS AT THE MCINTYRE
PORCUPINE

Mesh	Symons feed, per cent	Symons dis- charge, per cent	Screen feed, per cent	Roll feed, per cent	Roll dis- charge, per cent	Mill-bin feed, per cent
+6-in. ring.....	16.1					
+4-in. ring.....	7.8					
+2-in. ring.....	17.5					
+1½-in. ring.....	5.7					
1.050 (Tyler).....	3.4	1.5	0.8	1.3	0.3	
0.742 (Tyler).....	7.1	8.7	4.2	6.8	1.9	
0.525 (Tyler).....	6.4	17.2	7.3	10.4	2.9	
0.371 (Tyler).....	5.1	16.3	8.3	13.5	5.1	
3 mesh.....	4.4	9.8	9.8	16.1	11.9	
4 mesh.....	3.4	6.8	20.6	31.0	29.0	2.2
6 mesh.....	3.4	7.1	16.5	16.6	20.7	14.6
8 mesh.....	1.8	4.2	8.7	2.1	6.2	15.7
10 mesh.....	2.0	5.2	4.9	0.5	5.6	15.2
14 mesh.....	2.2	2.0	2.9	0.2	2.7	8.6
20 mesh.....	1.5	2.9	2.6	0.1	2.5	8.1
28 mesh.....	1.7	2.7	2.1	2.2	6.6
35 mesh.....	1.4	2.7	1.8	1.7	5.5
48 mesh.....	1.4	1.8	1.4	1.3	4.3
65 mesh.....	1.0	1.8	1.3	0.5	0.9	3.1
100 mesh.....	1.2	1.5	1.1	0.9	3.0
150 mesh.....	0.7	1.0	0.8	0.7	1.9
200 mesh.....	0.4	0.7	0.6	0.4	1.7
-200 mesh.....	4.4	6.1	4.3	0.9	3.1	9.5
	100.0	100.0	100.0	100.0	100.0	100.0

NOTE.—Screen feed is Symons discharge plus roll discharge. Roll feed is screen oversize.

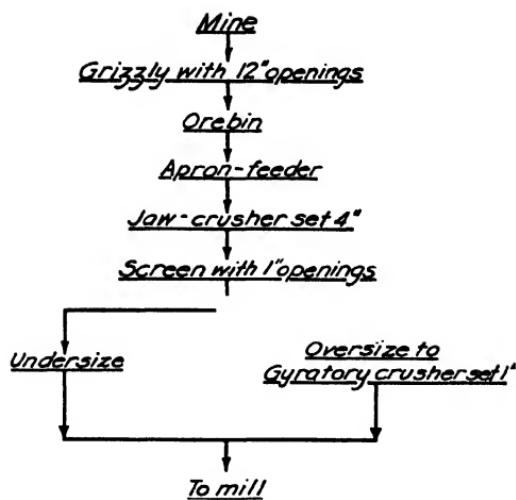


FIG. 13.—Two-stage crushing by jaw and gyratory crushers, with screen in open circuit.

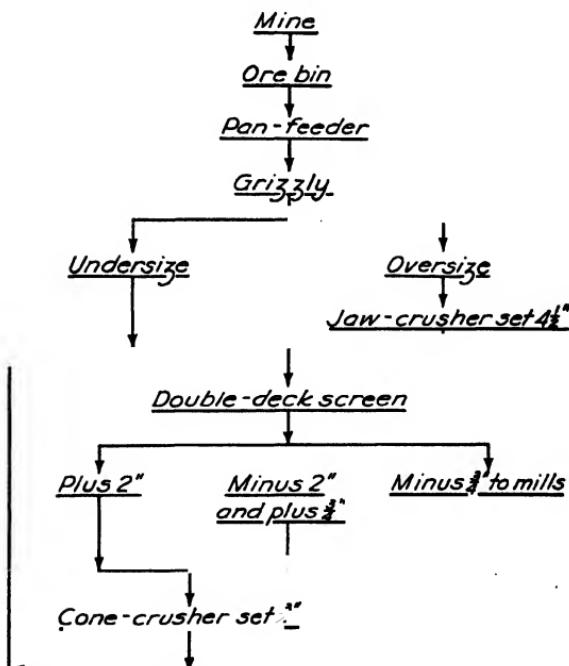


FIG. 14.—Two-stage crushing with grizzly and screen, the latter in closed circuit with second crusher.

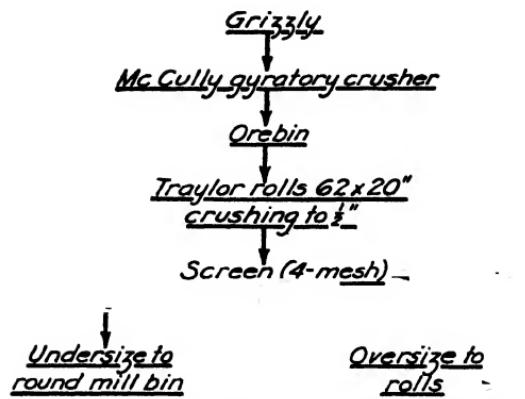


FIG. 15.—Coarse crushing at the Coniaurum, Ontario.

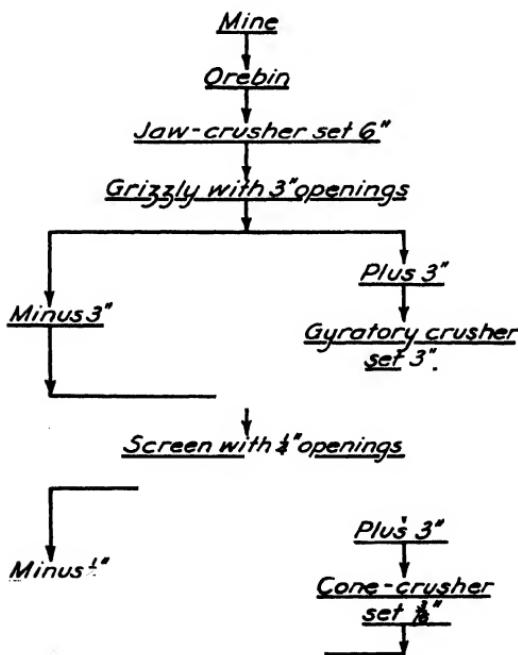


FIG. 16.—Three-stage crushing with grizzly and screen in open circuit.

It costs 10.7 cts. per ton to do this work. Tube mills in closed circuit with Dorr classifiers further reduce the ore, eventually, all through 200 mesh.

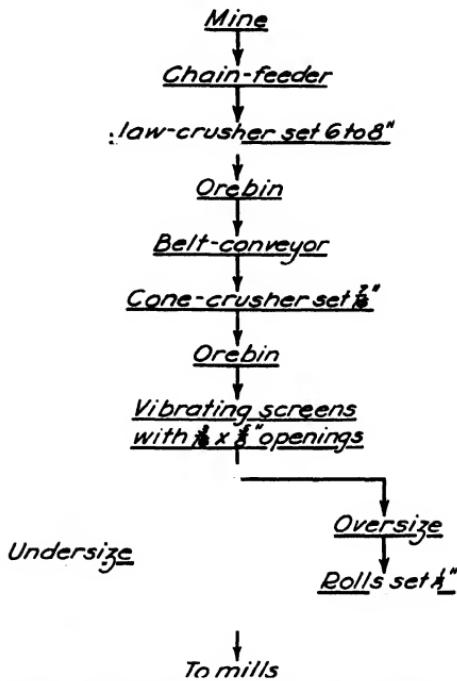


FIG. 17.—Three-stage crushing with jaw crusher, cone crusher, rolls and screens in open circuit.

The cost of coarse crushing in four stages to $\frac{1}{2}$ in. at the Hollinger mine, Ontario, is 0.78 ct. per ton for the underground jaw crusher; 1.28 cts. for the size- $7\frac{1}{2}$ gyratories; 0.88 ct. for the size-5 gyratories; 1.57 cts. for the rolls; 0.61 ct. for screening; 0.85 for conveying; 0.37 ct. for collecting dust and 2.27 cts. for conveying to mill bins. Total coarse crushing cost—8.61 cts.

CHAPTER IV

SORTING

When ore in mining becomes diluted by country rock because of narrow slopes or because of weak wall rock, or when it naturally contains barren material which is distinguishable from the ore, sorting and rejecting the worthless rock may be economical. This is done in a casual manner at some small mines and on a large scale at others, as at Alaska-Juneau (40 per cent rejected), or covering whole districts, as at Kolar, India (16 per cent), and on the Rand (15 per cent). Sorting at a gold or silver mine may mean its existence, as at the Alaska-Juneau, or it may result in lower costs because less ore is crushed and treated to recover the same amount of gold, as at Cripple Creek, Kolar and on the Rand. Sorting may rid an ore of such worthless material as schist which causes trouble in settling and filtration, as at Kochkar, southern Russia.

The place at which sorting shall be done is mainly dependent upon the size of the mine-run ore. If the ore is not in too big chunks, the sorting belt may be placed below the grizzly or trommel to receive the oversize, after it has been sprayed with water. The grizzly undersize is transported direct to the storage bin. If mine-run ore is in too large pieces, it should be broken to 4- to 7-in. size, sprayed and then fed to the sorting belt. The wash water containing the fine material sometimes carries enough gold to be worth saving; therefore provision should be made for sampling and assaying it also for its proper disposal.

EXAMPLES OF SORTING

Several examples of sorting follow:

Alaska-Juneau Practice.--The gold of the Alaska-Juneau has been deposited on the outside of the quartz or in fractures in the veins, associated with muscovite, ankerite and some pyrite, according to Livingston Wernecke, consulting geologist. In this respect it is similar to the gold deposits of the Mother Lode of California, in Nova Scotia, at the Howey, Ontario, and at Bendigo, Australia. W. P. Scott, the mill superintendent, reports that the ore is well-suited for hand sorting. Almost all of the valuable minerals occur in the quartz or at the contact of the quartz and the gabbro. No special skill is required in sorting, this involving selection of the white quartz, or pieces with adhering quartz, from the darker gangue material. The

metagabbro, the principal gangue constituent, is hard and tough.

Sorting is done at two points. The first is on 4-in. rock which comes from grizzlies under the jaw crushers. Four men each remove 70 to 80 tons of ore per shift from the 42-in. belt conveyors, which travel at 100 ft. per min. The waste remains on the belt and is taken to the dump. From 1000 to 1200 tons of coarse rock is sorted out as ore in 24 hr. This is crushed, and eventually an undersize is fed on to two 42-in. belts traveling at 150 ft. per min. Four men at each belt take out 20 to 25 tons of ore per man shift, leaving the rock on the belts. The ore sorted and screened at this point ranges from 500 to 700 tons per shift. As much as 57 per cent of 13,000 tons of rock sent to the mill has been discarded, but the nature of the rock has changed, and reject now amounts to 40 per cent of about 11,000 tons trammed daily.

SORTING AT ALASKA-JUNEAU

Product and Value	Quantity
Rock extracted, tons.....	4,085,960
Assay, cents per ton.....	98
Rock sorted out (40%), tons.....	1,619,128
Assay, cents per ton.....	17
Ore milled, tons.....	2,466,832
Recovery, dollars per ton.....	1.27
Mill tailing, cents per ton.....	24
Cost of sorting, cents per ton (1931).....	3.27

As shown in the accompanying table, sorting at Alaska-Juneau has raised the grade of the ore and saved much wasteful and costly milling. This is good work by the sorter, especially as the rejected rock assays 7 cts. less than the mill tailing. During 1931 each sorter rejected 68 tons of rock per day.

Cresson Practice.—The mine of the Cresson Consolidated Gold Mining and Milling Company at Cripple Creek, Colo., produces a sulphotelluride ore. The tellurides are sylvanite and calaverite, which occur in small quartz and fluorite veins, also disseminated in a basaltic breccia. The gangue is mainly Cripple Creek breccia. According to A. H. Bebee, superintendent, the ore when dumped into bins assays 0.28 oz. per ton in gold. A daily average of 35 per cent of the ore is sorted out, but the range is zero to 75 per cent. The reject carries 1.0 dwt.,

and the ore shipped 8 dwt. Each man sorts 15 tons a shift at a cost of 35 cts. per ton shipped.

Howey Practice.—At the Howey mine, Red Lake, Ont., the ore body is comprised of a series of quartz veins in a sheared and fractured porphyry dike, according to J. L. Ramsell, chief engineer, in *C.M.J.*, October, 1934.

The sulphide content of the ore averages about $2\frac{1}{2}$ per cent and is composed mainly of pyrite with minor amounts of sphalerite, galena and a little chalcopyrite. Ankerite and tourmaline are common. Howey quartz-vein distribution is remarkably similar to that of the Alaska-Juneau, and similar problems in ore definition and extraction have resulted.

An outstanding feature of ore treatment at the Howey is its hand sorting before crushing, according to R. S. Sullivan, mill superintendent, in the journal cited. As a result of this operation, a low-grade material, amounting to 18 per cent of the ore hoisted from the mine, is discarded. The cost of sorting and disposing of waste rock during the first six months of 1934 amounted to 25.7 cts. per ton discarded or 4.3 cts. per ton hoisted.

Experience at this mine has shown that any rock not carrying quartz or sulphides is invariably low-grade material and can safely be discarded as waste. The gangue is easily discernible by reason of its dull-black appearance when wet. No sledging or cobbing is practiced, largely because the resultant gain would be more than offset by the loss in time and tonnage due to the interruption of the sorting rate. Thus it is necessary to rely mainly on the judgment of properly trained sorters.

The size of material most suitable for sorting ranges from 6 to 14 in., and the sorting rate is 3 to 7 tons of waste per man-hour discarded with a long-term average of 4.5 tons. Present equipment does not permit the economical sorting of minus 4-in. material, because the number of moves made by sorters to remove small pieces is so great that their sorting ability becomes extremely low. Even with more suitable equipment, it is quite probable that in handling the smaller material, the sorting rate would fall below 2 tons per man-hour.

Ore is delivered to the crude-ore bin of 250 tons drawoff capacity and varies from fine material to slabs that will pass the 14-in. mine grizzlies. The flow of rock from the mine bin to the sorting table in the crushing plant is controlled by horizontal

and vertical steel gates set below and in front of the chute and operated by electric control levers from the control board at the jaw crusher. Each gate, consisting of heavy steel plate reinforced by 30-lb. rails, is guided by 7-in. channels and is operated by a 1-hp. motor through a series of pinions and spur gears.

As soon as the rock reaches the table it passes under a heavy spray of water which removes the slime, permitting the waste to be distinguished easily from the ore. The sorting table is a link-belt, manganese-steel pan conveyor with 19-ft. centers, 60 in. wide with stationary guides consisting of 6- 6-in. timber, lined on the inside edge with $\frac{1}{4}$ -in. lapped steel and on the top with $\frac{1}{8}$ -in. steel plate. This table, traveling at a speed of 8 ft. per min., is driven by a 10-hp. motor through a 4-in. belt, pinions and spur gears and has given three years of excellent service requiring only minor repairs.

A small drag-chain classifier, set directly below the table and operated from the table driven shaft, discharges all the fine material washed down by the spray, to the 24-in. conveyor. Four sorters, two on each side of the table, stationed 3 to 4 ft. apart transfer the waste to bins directly behind them. The reject is drawn from the waste bins to 2-ton dump cars which are hauled away to be emptied. Sorting results at the Howey mill for the first half of 1934 are shown in the accompanying table.

SIX MONTHS' SORTING AT THE HOWEY MILL

Operation	Quantity and Value
1. Ore milled and sorted, tons.....	239,488
2. Ore milled, tons.....	199,190
3. Discarded by sorting, tons.....	40,298
4. Value of ore milled and sorted, per ton.....	2.50*
5. Value of ore milled per ton.....	2.93
6. Value of material sorted out per ton.....	0.41
7. Tailing loss per ton.....	0.22
8. Net loss (6 and 7) per ton.....	0.26
9. Increase in mill heads due to sorting.....	0.43

* Gold at \$35 per oz.

During $4\frac{1}{3}$ years, 153,377 tons of rock was rejected from 1,250,156 tons of ore milled and sorted. The waste averaged 49 cts. per ton, and the mill heads were increased in value by 41 cts.

Rand Practice.—The reason for sorting on the Rand is better understood when it is said that the "reef" material is a conglomerate composed of various sizes of barren quartz pebbles about grape size, cemented together, the cementing material containing the gold and minerals, lying in barren quartzite shale. As stopes are narrow, much waste is mixed with the ore. Some of this is sorted out underground, but 15 per cent of that hoisted to the surface is rejected, and at several plants nearly double this amount is discarded. The waste assays up to $\frac{3}{4}$ dwt. per ton. Natives do this work well and can sort down to $1\frac{1}{2}$ -in. size. Where there is enrichment at the reef-country rock contact, particularly at the shale wall, or when gold is in scattered carbon particles, sorting is more difficult. The cost of crushing the ore, one-half to two-thirds through 1-in. screen, and sorting ranges from 3 to 5 pence (6 to 10 cts.) per ton milled. Selected pebbles and quartzite serve well as grinding media for tube mills.

As a rule, on the Rand, the fine material is screened out before the ore is washed and sent to the sorting belts. This screening is done for the most part on grizzlies and vibrating screens. Washing is done in revolving trommels. At a few of the plants all of the ore is washed and screened before sorting. The fine ore is sent direct to the stamps or other grinding machines. Waste rock is scraped into pockets which lead to bins, these in turn discharging on to a waste belt or into cars or trucks.

At the East Geduld mine, sorting underground is done at a station 2930 ft. below the surface. The ore from two sets of tipplers or dumpers first passes over a 4-in. grizzly. The undersize goes to a bin at the shaft; the oversize is fed on to two 36-in. sorting belts, being well-washed in the inclined section of the belt. The sorted ore drops into the ore pass, joining the undersize from the grizzly; the waste is dropped into another bin and is then distributed to the mine workings as filling. About 4000 tons of waste per month is handled underground.

Sorting on the surface at East Geduld consists of washing and screening the ore from the primary crushers. Here it is divided into three sizes: (1) Minus 7 in. and plus $3\frac{1}{2}$ in.; (2) minus $3\frac{1}{2}$ in. and plus $1\frac{1}{4}$ in.; (3) minus $1\frac{1}{4}$ in. The ore is thoroughly washed, the washings gravitating partly to a Dorr classifier and partly to two 4-in. Wilfley sand pumps, from which the

materials go direct to the tube-mill circuit. The three sizes of washed ore are fed on to three 36-in. belts for sorting. The plus $3\frac{1}{2}$ and minus 7-in. material goes to a tube-mill pebble bin; if this is full, the ore is sent to the secondary crusher bin. A similar arrangement is available for the plus $1\frac{1}{4}$ - and minus $3\frac{1}{2}$ -in. product. The cost of sorting and crushing is $6\frac{1}{2}$ pence or 13 cts. per ton.

Of the 20,327,000 tons of material broken in the mines of the Central Mining-Rand Mines group in 1933, according to T. K. Prentice in the *Bul., I.M. and M.*, April, 1935, 13.79 per cent was waste packed underground, 4.97 per cent was waste hoisted and sent direct to the dump, 8.84 per cent was waste sorted out in the reduction plant and 72.40 per cent was ore milled.

CHAPTER V

FINE GRINDING

Fine grinding is the last step in the reduction of an ore prior to concentration (gravity or flotation) or cyanidation. Practice varies, depending upon the type of ore and the amount of reduction required. In addition, some of the older properties continue with methods that perhaps are not considered the best in light of recent improvements but that cannot be economically changed because of capital outlay.

Present grinding practice is closely linked with classification, so that some overlapping of subject matter occurs in Chaps. V and VI.

In this chapter some of the theory of grinding, different types of equipment and flow-sheets are discussed. Concentrate regrinding is discussed in Chap. IX.

EQUIPMENT

Most of the tonnage milled today is ground in one of the following types of equipment or a combination of two or more: ball mills, tube mills, rod mills and stamps. Chilean mills and Huntington mills are still used in a few isolated cases.

BALL AND TUBE MILLS

All types of mills that consist essentially of rotating cylinders with flat or conical ends with balls, pebbles or rods used as grinding media are included here.

Ball Mills.—The term "ball mill" is usually used to refer to a cylindrical mill whose length is less than, equal to but not much greater than its diameter. It was initially developed for relatively coarse grinding, but by using it in closed circuit with a classifier its use has been extended for fine grinding.

Ball mills have shells of cast iron or steel plates and are carried on hollow trunnions. Ore is fed through a scoop, drum type or combination feeder at one end and is discharged from the opposite trunnion.

Ball mills may be arbitrarily classified into two types, according to the method of pulp discharge. In high-level or overflow mills the pulp level builds up until it overflows and discharges

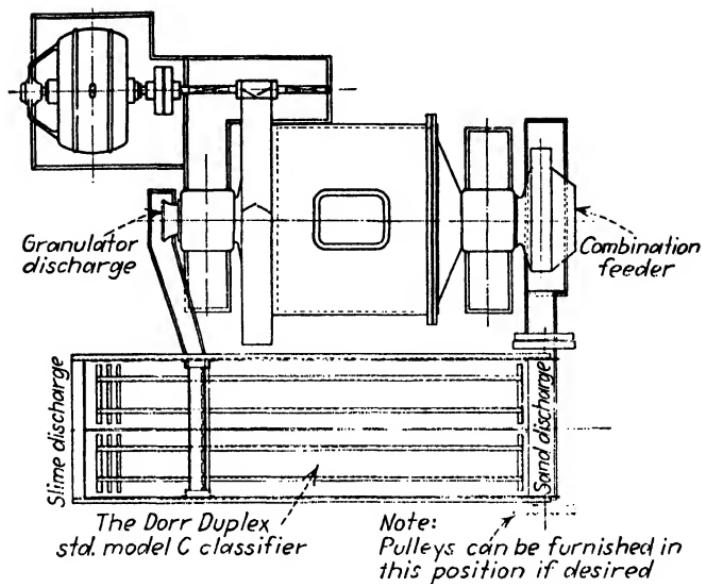


FIG. 18.—Allis-Chalmers ball granulator in closed circuit with Dorr duplex classifier

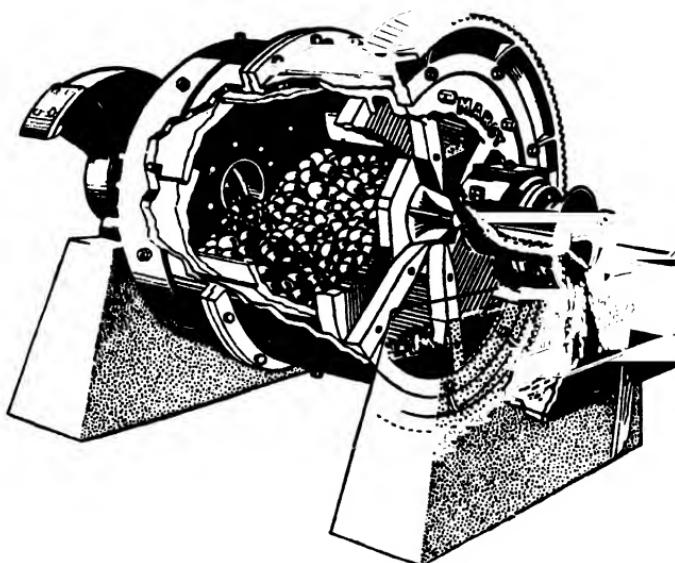


FIG. 19.—Sectional view of Marcy ball mill in operation.

through the trunnion. High-level discharge mills are made by a large number of manufacturers throughout the world. Low-level mills are typified by the Allis-Chalmers and Marcy (see Figs. 18 and 19) grate-discharge mills. The discharge end is fitted with grates; between the grates and the end of the mill are radial lifters which act as a pump to lift the discharge to the hollow trunnion. Drive is by spur or herringbone gear, direct connected or belt driven.

Ball mills are built in sizes ranging from small laboratory mills to a present maximum of $10\frac{1}{2}$ ft. diameter by 8 ft. long, the latter requiring close to 500 hp.

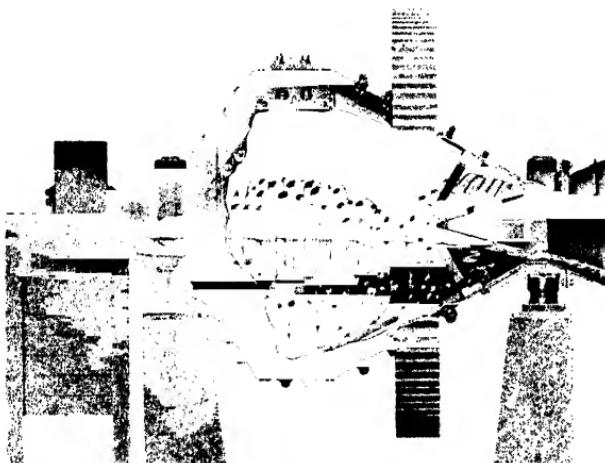


FIG. 20.—Hardinge conical mill fitted with paddle grate.

Liners are usually of manganese steel, of chrome steel or white iron, 3 to 6 in. thick. Corrugated and shiplap construction is commonly used to increase the grinding action.

Conical Mills.—The Hardinge mill (see Fig. 20) differs from most ball mills in that conical ends are added to the cylindrical portion of the mill. The cone at the feed end has a larger open angle than that at the opposite end. Its makers state that the large balls concentrate near the feed end of the mill where the coarsest ore collects, and the smaller balls act on the finer ore.

Rod Mills.—Rod mills (see Fig. 21) follow the general dimensions of tube mills with diameters from 3 to 6 ft. and lengths from two to three times their diameter. They differ from ball mills

in that steel rods 3 or 4 in. shorter than the mill length inside the liners are used as grinding media. Rod mills are often run on tires and rollers instead of trunnions or on one trunnion and one tire and set of rollers.

Low-level discharge is obtained on Marcy rod mills by having a beveled annular ring at the discharge end. A stationary steel door fits close to this beveled ring and serves to hold the rods in the mill while pulp discharges between the mill and the door.

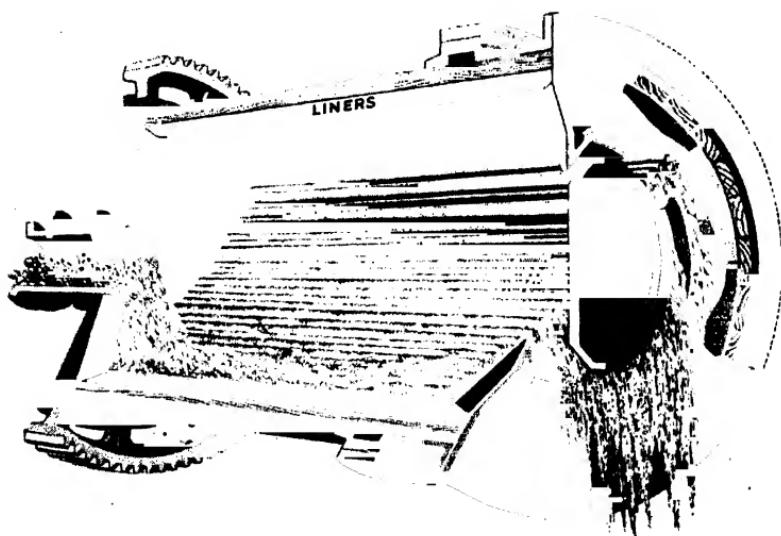


FIG. 21.—Sectional view of Marcy rod mill in operation.

Tube Mills.—The distinction between tube mills and ball mills is not so marked as their names indicate. Mills from 4 to 6 ft. in diameter and from 16 to 22 ft. long are usually termed tube mills. This was the first type of rotary mill for metallurgical purposes. Because of the necessity of completing the grind during one passage (open circuit) of the ore through the mill, it was built with a large-length diameter ratio. The tube mill is still largely used in South Africa and to some extent in North America for fine grinding generally following some other primary mills.

Tube mills are usually supported on hollow trunnions, the feed entering through a feed scoop at one end and discharging through

the other. Drive is by a large gear fitted over the mill shell. Various types of liners are used, as in ball mills.

Flint pebbles and hard waste rock were first used as grinding media, but present practice is to use small steel balls (less than 2 in.) or, as in South Africa, a mixed steel ball and rock load.

GRINDING MEDIA

Steel balls ranging from $\frac{7}{8}$ to 6 in. in diameter are used. Tube mills are usually fed balls smaller than 2 in., whereas 5-in.



FIG. 22.—Interior of Sylvanite plant, Kirkland Lake. Ball mill in circuit with straight classifier and tube mill in circuit with bowl classifier.

balls are most commonly used for coarse grinding. The initial charge is made up of mixed sizes, and thereafter only the largest size is fed to the mill. A grinding capacity much higher in tube mills than with pebbles is of course obtainable with steel media.

Rods used are of carbon steel $1\frac{1}{2}$ to 4 in. in diameter and containing 0.8 to 1.0 per cent carbon. Mild steel rods have a tendency to bend after wearing down.

Approximate ball loads can be estimated by assuming 225 lb. per cu. ft. of ball volume and a total load equivalent to 40 to 45 per cent of the mill volume.

LINERS

All rotary mills must be fitted with some kind of replaceable liners. Chrome steel, manganese steel and white iron are usually used. Shapes designed to give a corrugated or shiplap surface to the interior of the mill are often used to prevent slippage of the ball load. Pocket liners are also common. These liners have pockets in which the balls become lodged to form the wearing surface.

Rubber liners have been tried in some cases but are not common. Silex lining, with flint pebbles for grinding media, is sometimes used where iron contamination must be avoided.

OLDER GRINDING EQUIPMENT

Stamps.—Although the stamp mill could be classified as a fine crusher, it is included in this chapter, as its usual duty corresponds approximately to that of a primary ball mill.

Stamp mills were built to parallel the operation of a mortar and pestle, working continuously and on a large scale. Ore is fed into a mortar and is crushed by the dropping of the stamp on a die at the bottom of the mortar. The crushed ore discharges through a screen in the side of the mortar.

The shoe that forms the wearing surface on the dropping stamp is attached to a steel stem and is replaceable. A common cam shaft activates usually five stamps in a battery. Weights vary from 850 to 1500 lb. in the United States and from 1500 to 2000 lb. in South Africa.

Chilean and Huntington Mills.—These two types of mill are practically obsolete. In these mills rollers driven from a central gear-driven spindle revolve around a pan. In the former the rolls crush against a ring in the bottom of the pan, and in the latter centrifugal force holds the rollers against the ring at the side of the pan. Chilean mills are still used at the Golden Cycle for grinding roasted ore.

RECENT DEVELOPMENTS

Hadsel Mill.—The Hadsel mill (see Fig. 23) is a new type of grinding device now in the demonstration stage. A large wheel, up to 24 ft. in diameter by 4½ ft. wide, is fitted with internal buckets. As the wheel revolves, the buckets lift the rock

to the top of the mill where it is dropped on to stationary breaker plates. The ore is thus crushed by its own impact on the plates. Ore as large as 12 in. is fed to the mill.

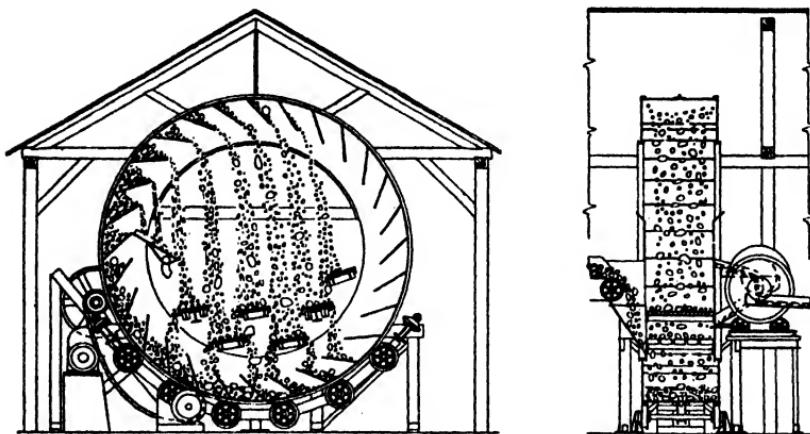


FIG. 23.—Principle of operation of the Hardinge-Hadsel mill.

THEORY AND PRACTICE BALL AND TUBE MILLS

Grinding Action.—The grinding that takes place in mills of this type is usually ascribed to two actions, impact and attrition, although some authors do not believe that a sharp line of demarcation can be drawn between the two actions.

In rod mills there is line contact between the rods, and the fines exert a greater cushioning effect than when balls are used. As a result, a rod-mill product usually contains a greater percentage near the limiting size, with less extreme fines than ball or tube mills.

Capacity.—The capacity of a given mill will vary over rather wide limits depending on conditions given in the accompanying table. An accurate estimate of capacity can be made only by an engineer familiar with the proper evaluation of these factors:

1. Character and size of feed.
2. Character and size of product.
3. Size and number of balls or rods.
4. Speed of mill.
5. Type of liners.
6. Open- or closed-circuit grinding.



FIG. 24.—Ball mill in closed circuit with Dorr classifier.

7. Ratio of circulating load if the grinding is in closed circuit.
8. Percentage moisture in pulp.

For rough estimating purposes the accompanying table gives approximate capacities to 48 and 100 mesh for several size mills. Connected horsepower is also shown. These figures are

APPROXIMATE BALL-MILL CAPACITIES AND POWER REQUIREMENTS

Size of mill, feet	48 mesh	100 mesh	Horsepower connected
4 by 3	25	10	20 to 30
5 by 6	85	35	60 to 75
7 by 6	320	125	150 to 175
8 by 6	500	200	200 to 225

for what would normally be considered average siliceous ore crushed through $\frac{3}{4}$ in. and for nominal circulating loads of 2 or 3 to 1. For any special problem the manufacturers should be consulted.

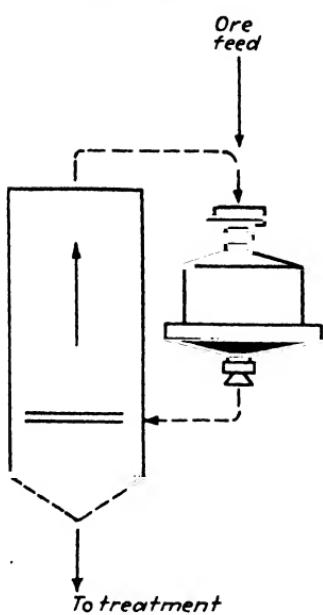


FIG. 25.—Single-stage, medium-fine grinding with classifier in closed circuit with ball mill.

Open- and Closed-circuit Grinding.—When the tube mill was first introduced, grinding was done in open circuit; that is, the ore was ground to pass the limiting screen size by one passage through the mill. Early experiments showed that about 50 to 75 per cent of the grind took place in the first quarter of the mill and 80 to 95 per cent was finished in the first half. In this manner a large part of the ore is ground much finer than is necessary, and mill capacity is relatively very small.

For this reason open-circuit grinding is now seldom used. A classifier (see Fig. 27) is used to remove oversize from the mill discharge, and no attempt is made to finish the grind by one passage through the mill, but every effort is

made to remove finished material as soon as it is finished, thus reducing overgrinding and preventing the fines from hindering the grinding action on yet unreduced particles. In this way the tonnage that a given mill will grind is much greater than it is possible to grind in open circuit.

The different types of classifiers used are discussed in the chapter on classification. In general, mechanical classifiers that have a large sand-raking capacity are better because higher circulating loads are possible.

There is some controversy in the literature as to the definition of ratio of circulating load. The term as used by most millmen is the ratio of sand tonnage returned to the mill to the tons of original feed.

If the mill-classifier circuit is fed in the classifier instead of in the mill, the sand contains oversize in the original feed as well as oversize from mill discharge, and thus the definition is not entirely accurate.

The ratio of circulating can be calculated from screen analyses by using the following formulas:

1. When the raw feed enters the mill directly:

$$\text{Circulating load rates} = \frac{d - o}{s - d}$$

d = cumulative percentage on any mesh in the mill discharge.

o = cumulative percentage on same mesh in the classifier overflow.

s = cumulative percentage on same mesh in the classifier sand.

2. When the raw feed enters the classifier:

$$\text{Circulating-load ratio} = \frac{f - o}{s - d}$$

f = cumulative percentage on any mesh in the raw feed.

d, s, o are the same as in 1.

The usual procedure in using these formulas is to calculate the circulating-load ratio for several size screens and discard any that appear out of line, averaging the remainder.

Tests have shown that the capacity of a mill to grind to a given mesh increases greatly by changing from open- to closed-circuit grinding with a small circulating load and continues to increase at a lower rate with increase in the circulating load.

By using wide classifiers with high raking capacity, circulating-load ratios are now being carried as high as 10 to 1 or higher. The direct result of the increased capacity is reduced power, liner and grinding media consumption per ton of finished ore.

Mill Speed.—The speed of grinding mills is a function of the internal radius and is usually 50 to 80 per cent of the critical

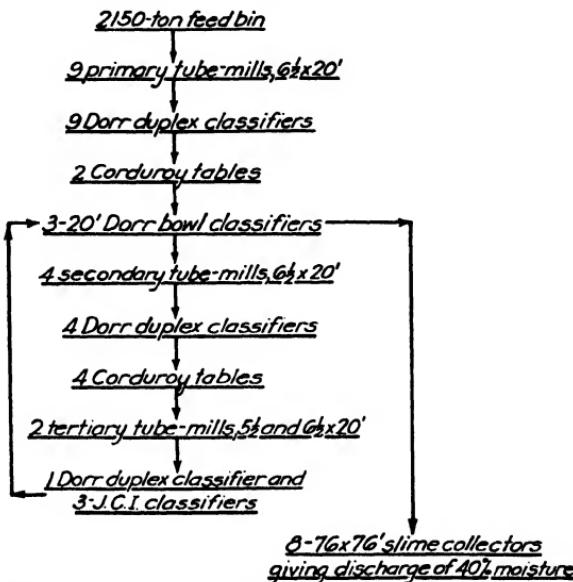


FIG. 26.—Flow-sheet of New State Areas tube-milling plant.

speed, the speed at which material starts to cling to the mill as it revolves. This speed

$$N = \frac{76.50}{\sqrt{d}}$$

where N = r.p.m.

d = diameter

inside the liners in feet.

Moisture in the Pulp.—Usual practice is to keep the moisture in the mill at from 40 to 25 per cent. Decreased moisture reduces slippage and allows balls to be carried higher in the mill. As a result, capacity increases slightly. In general, moisture should be increased with an increase in slime content. A consistency is desirable which causes the pulp to coat and cling to the balls

in a layer, but that is not so great as to prevent the pulp from flowing readily through the mill.

FLOW-SHEETS

There are many types of flow-sheets in use today (see Figs. 26, 27 and 28). However, the tendency in new mills is to crush relatively fine ($\frac{1}{4}$ to $\frac{1}{2}$ in.). For grinds coarser than 48 mesh, single-stage ball mills in closed circuit with classifiers are used; but when a finer product is desired, two stages of ball mills in closed circuit with classifiers is usual. In small mills efficiency must necessarily be sacrificed to some extent by capital requirements, and even greater reductions are justified in a single-stage grinding unit.

With the large classifiers used for high circulating loads it is quite often necessary to use some kind of auxiliary device to complete the closed circuit. A large auxiliary scoop lifting the mill discharge to the classifier has been successful.

Concentrate cyanide pulp

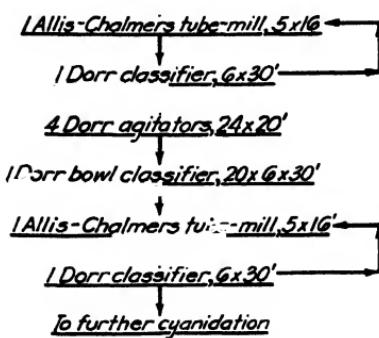


FIG. 28.—Flow-sheet of McIntyre Porcupine tube-milling plant.

is summarized by T. K. Prentice in *Bul., I.M. and M.* 367, April, 1935, as follows:

During the last five years, three entirely new plants have been erected on the Witwatersrand field, namely, East Geduld, Daggafontein, and

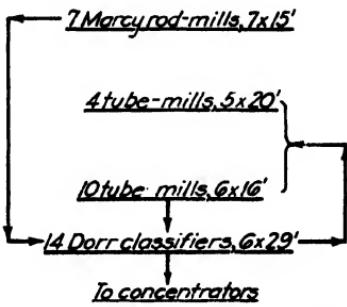


FIG. 27.—Flow-sheet of Hollinger tube-milling plant.

Rod mills are used to some extent for relatively coarse grinding, but the trend of recent practice has been toward ball rather than rod mills.

In North America stage grinding is often carried out with ball mills as primary grinders and tube mills for fine grinding in the second stage.

RAND PRACTICE

Recent, compared with older, grinding practice on the Rand

Sub-Nigel, and one of these incorporated either stamps or sand-treatment plant. Modern high-speed gyratory crushers were installed at each for preparing the feed for the tube mills, which are mostly 6 ft. 6 in. in diameter by 20 ft. in length, though there are some which measure 8 ft. by 16 ft. In each plant, the "mines fines," mostly—1 in. square mesh are by-passed to the tube-mill bins without preliminary washing. In two of the plants tube milling is performed in a cyanide-solution circuit, and at the other in a water circuit. In all of them Dorr classifiers of the straight drag and of the bowl types are used exclusively for classification, and only the Sub-Nigel plant, which, incidentally, mills the richest ore on the Rand, has installed corduroy tables for the removal of gold from the tube-milling circuit, for recovery by amalgamation; the others depend entirely on cyanidation for the total gold recovery. At East Geduld air lifts are used to elevate the tube mill effluent to the classifiers; and at the other two plants centrifugal pumps are used for this purpose. Three stages of tube milling were incorporated at East Geduld and Daggafontein, and two stages at Sub-Nigel. . . .

On the Witwatersrand there are 38 large reduction plants with an average milling capacity of approximately 87,000 tons per month each. Of these, the last 6 to be erected did not incorporate either stamps or sand-treatment units, and in 1930 the Springs Mine abandoned sand treatment, but the remaining 31 still function satisfactorily and economically with the inclusion of both. The principal benefit to be gained by not incorporating stamps and sand-treatment units in the flow-sheet of a new plant is in the matter of capital cost. A plant having a capacity of 50,000 tons per month with stamps and sand-treatment equipment would cost on the Rand approximately £400,000, against £300,000 for a modern crusher tube-mill all-sliming plant.

The cost of grinding depends on the degree to which comminution is carried, but on the basis of the cost per ton of —100¹ or per ton of —200 produced, statistics show that the inclusion of stamps does not appreciably adversely affect the grinding cost. In the Central Mining-Rand Mines group it has been found that the cost of producing one ton of —100 product by the crusher-stamp and tube-mill combination is 20.6 pence compared with 19.9 pence by the modern crusher tube-mill combination and that the costs per ton of —200 produced are 29.7 pence and 28.3 pence by each respective method. The electric energy required to produce a ton of —200 is approximately 27 kw.-hr. and is the same for either method of reduction. Thus it may be accepted that the stamp mill is still performing useful and economic work and that,

¹ All screen gradings in this paper refer to Tyler screening unless otherwise stated.

although it has been suggested, particularly by crusher-machinery salesmen, that it would be profitable to the mines to scrap the existing stamps, the capital expenditure involved in the purchase of the crushers and tube mills to replace the stamps would be difficult to justify.

In the group of mines previously referred to, the cost of cyaniding in an all-sliming plant amounts to 12.3 pence per ton milled, compared with 13.6 pence per ton milled in the other plants incorporating sand and slime treatment. Thus, there is a saving of 1.3 pence per ton milled in favour of treatment by the all-sliming method, and, provided the increased extraction is sufficient to cover the cost of the increased fineness of grinding necessary to enable the ore to be treated as a one-pulp product, then all-sliming is economically justified. From the mechanical aspect the degree of grinding necessary for satisfactory treatment by all-sliming methods is about 95 per cent of -100 or 70 per cent of -200. The existing all-sliming plants are all situated on the Far East Rand where fine grinding is essential for a reasonable recovery, and there never has been any doubt about the advisability of adopting fine grinding there; but the Central and West Rand ore is less refractory, and the question whether all-sliming should be applied to that class of ore has long been a debatable subject. However, there are two plants at present being designed for the West Rand area, and both are incorporating all-sliming, so that it is reasonably safe to prophesy that no future large plant will incorporate sand treatment unless sand is required for underground roof support; nor will any future new plant incorporate stamps.

GRINDING AND CLASSIFICATION AT RANDFONTEIN ESTATES

F. Wartenweiler gives us the following flow-sheet at this great property: In the crushing plant, 800 tons of waste is sorted out daily. This plant delivers to the 600 stamps 2.9 per cent of plus 2-in. ore, 33.1 per cent of plus 1 in., 15.8 per cent of plus $\frac{3}{4}$ in., 18.0 per cent of plus $\frac{1}{2}$ in. and 30.2 per cent of minus $\frac{1}{2}$ in. The stamp pulp is pumped to spiral classifiers which are in closed circuit with 8- by 16-ft. tube mills. These classifiers are $7\frac{1}{2}$ and 9 ft. in diameter and make 13 r.p.m. In this circuit are corduroy tables. The critical mesh of this primary grinding is 100. The sand product amounts to 6069 tons. The overflow from these classifiers is next pumped to the secondary grinding circuit, which consists of 7- and 8-ft. spiral classifiers, $5\frac{1}{2}$ by 22-ft. tube mills, corduroy tables and cone classifiers. The sand product amounts to 1968 tons. These cones receive the spirals' overflow, the underflow of the former returning to the secondary

circuit, and the overflow (81 to 84 minus 100 mesh) going to pumps which elevate it to six 23-ft. Dorr bowl classifiers. These give 35.21 per cent sand and 64.79 per cent slime.

SOUTH AFRICAN PRACTICE
(Duties and Other Data with Pebble-grinding Medium)

Tube mill	Plant	Load, tons	Tons,	Tons,	Horse-	Consumption	
			100	200	power	per day	
			mesh per day	mesh per day		Balls, pounds	Pebbles, tons
Pebbles ^b							
5 ft. 6 in. by 22 ft.	C	13 to 14	157.0	120.0	146	...	30
5 ft. 6 in. by 22 ft. ^a	B	13 to 14	120.0	92.0	132	...	17
6 by 16 ft. ^a	B	13	140.0	105.0	147	...	25
6 ft. 6 in. by 20 ft.	B	15 to 16	165.0	128.0	235	...	35
8 by 16 ft. ^c	F	Primary mills	120.5	238	...	53
		Tertiary mills	231.3	233	...	28

Composite Grinding Load of Pebbles and Steel Balls

		Balls	Pebbles				
5 ft. 6 in. by 22 ft.	C	5	11	196	131	160	300
6 ft. 6 in. by 20 ft. ^d	A	7	13	253	148	265	500
6 ft. 6 in. by 20 ft. ^e	B	6	13.3	271	190	263	420

^a Mine fines have been separated and do not pass this circuit.

^b Ore pebbles fed are 4 to 10 in. mean diameter.

^c Size of pebbles 4 to 10 in.; size of balls, 3 in. diameter.

^d Grinding run of mine ore after crushing.

^e Grinding screened fines from run of mine ore.

NORTHERN ONTARIO PRACTICE

In the chapter "Treatment of Simple and Complex Ores" complete descriptions of the more important Canadian mills are given. In general, it may be said that grinding is done in two stages, using ball mills or rod mills as primary grinding and tube mills for secondary grinding. Exceptions to these are the McIntyre Porcupine and Coniaurum, which crush very fine (4 mesh) and grind single stage in tube mills.

The primary mills at Dome and Hollinger are operated in open circuit, but all other mills are closed circuit with classifiers.

Many of the smaller mills grind small tonnage in single-stage closed circuits, even when it is necessary to reduce the ore to 100 to 150 mesh.

CHAPTER VI

CLASSIFICATION

The metallurgical advantage of fine grinding in the treatment of many ores by the all-sliming cyaniding process has of course been realized ever since the introduction of the process. An economical means, however, of attaining a finely ground product has been a gradual development to which improvements in grinding mills and classifiers have contributed the greatest advance.

The mill-classifier combination should be considered as a unit. The change from open circuit to closed circuit, the use of high circulating loads and the development of secondary and tertiary grinding circuits are notable advances in producing an improved product, generally resulting in a higher extraction of the valuable constituents of the ore. This has been accompanied by a reduction in unit costs through a lowered consumption of power and steel and by increase in capacity of the grinding mill.

The increased price of gold justifies the careful consideration of the metallurgist to the possibility of reducing (undissolved) gold loss in tailings by finer grinding.

DEFINITIONS

Classification as applied to cyanide plants is usually a combination mechanical-hydraulic operation which separates the solid constituents of a flowing pulp into two portions according to their respective settling rates. Usually it implies the removal of a finished product, termed "overflow," from a product requiring further grinding, termed "sand." *Pulp* means a uniform suspension of finely divided solids in liquids as applied to mechanical classification, which is the usual method of making a sand-slime separation in cyanide plants. *Overflow* is the comparatively finer, more slowly settling portion of the original pulp which is carried over the tailboard or lip of the classifier by the flow of water; *sand* is the comparatively coarser, more rapidly settling portion of the original pulp which is discharged from the classifier by the mechanical action of the rakes. *Selective classification* is classification that has for its object the concentration of the heavy constituent of the original pulp, generally the sulphide,

in the sand product, so that it may be ground finer than the lighter portion, generally the gangue.

Fine grinding is the size reduction of crusher or roll products. A mill may operate open circuit or closed circuit. *Open-circuit grinding* is a method of comminution that produces the desired reduction in particle size by a single passage of the material through a mill. *Closed-circuit grinding* is a method of comminution in which a partly finished mill discharge is separated by the classifier into a finished overflow product and an unfinished sand product which is returned to the mill for further grinding.

USES OF CLASSIFICATION

1. To make a size separation; for example, a sand-slime separation so that each product may be given a different treatment.
2. For closed-circuit grinding, so that the maximum size of particle escaping from the circuit may be limited, and so that the useful work done by the mill may be increased by enabling the mill to operate largely in reducing oversize rather than in overgrinding material already fine enough.
3. For differential grinding of the heavy mineral constituent and the gangue, by selective classification which concentrates the mineral in the sand product and causes it to be ground finer than the gangue before it can escape in the overflow product.

The desired function, together with the character of the ore and such factors as the tonnage to be handled, mesh of separation, etc., determines the selection of the proper size and type of classifier. The Dorr single-stage classifier and bowl classifier are in use for making separations from as coarse as 10 (see Fig. 29) to as fine (see Fig. 30) as 350 mesh. The first classifier is better suited for making coarse separations and for handling large circulating loads where relatively small overflow capacity is required. The bowl classifier, limited to 48 mesh and finer, with the double washing action gives a closer separation at the desired mesh and a cleaner sand product; permits, through the selection of the proper bowl size, any required relation between overflow and sand-raking capacity and is more suitable as a mineral concentrator for selective classification.

Both machines are capable of adjustment to cover considerable range of conditions. Adjustments in slope of classifier tank, rake speed and dilution of overflow pulp and, in the case of the

bowl classifier, additional adjustments of backwash water and bowl speed are available to obtain efficient results at any mesh of separation. In designing the machine, due consideration is

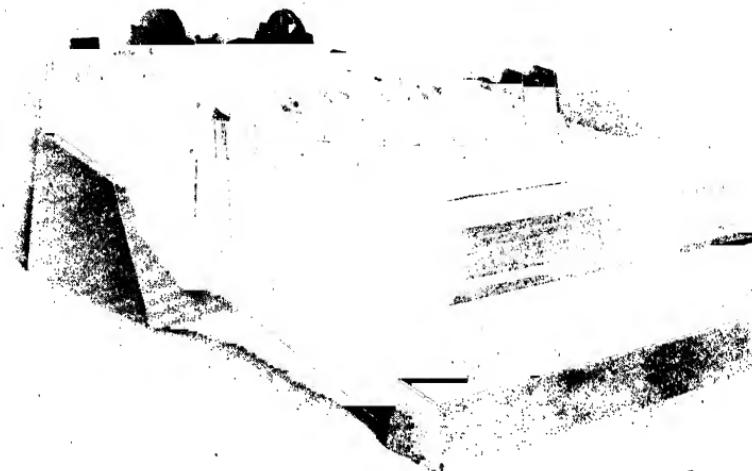


FIG. 29.—Dorr FX (heavy-duty) classifier for relatively coarse separations.

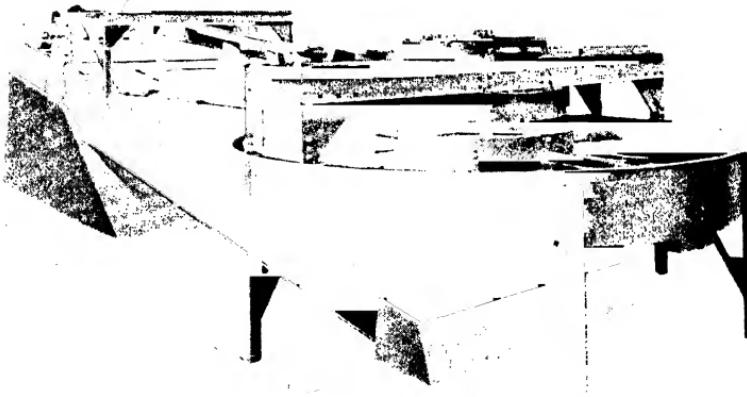


FIG. 30.—Dorr Turret-Bowl classifier, for relatively fine separations (Golden Queen).

given the results desired under specified conditions; final adjustments are made when the particular flow-sheet is placed in operation and are varied thereafter as the conditions vary.

Evolution of present fine-grinding flow-sheets:

Open-circuit Grinding.—The first application of the mechanical classifier was to open-circuit work with grinding mills in the cyanidation of gold and silver ores, for the purpose of producing two products—a slime-free sand for treatment by percolation and a sand-free slime for treatment by agitation and filtration or decantation.

Closed-circuit Secondary Mills.—Advances in the art of treating cyanide slimes and the reduced cost of fine grinding made possible by the early tube mills led to the use of the mechanical classifier as a means of controlling the fineness of the pulp leaving the grinding plant as feed to the all-slime cyanide treatment. Operating in closed circuit with the tube mills, the classifier not only controlled fineness more accurately but greatly reduced grinding costs.

Closed-circuit Primary Mills.—The adoption of two-stage fine grinding, due to a recognition that too great a size reduction in one mill with one ball charge was uneconomical, led to the use of the primary mill and subsequently a mechanical classifier in closed circuit with it. Reductions in grinding costs were again obtained, and in addition a convenient means was provided for apportioning the work between the two stages.

Intermediate Classification between Stages of Fine Grinding.—Another step was the interposition of a bowl classifier between the primary and secondary grinding circuits. The classifier operated in open circuit receiving as its feed the overflow from the primary circuit, overflowing material of finished size and discharging a clean sand product direct to the closed-circuited secondary mill. This step, however, is not generally desirable largely because of the difficulties met in controlling the succeeding classification stage in the absence of primary slime.

Finishing Bowl Classifiers.—Finishing the entire mill feed in a bowl classifier, with provision for regarding the sands in a separate or tertiary circuit, first proved attractive in the cyanidation of gold ores in which the values were chiefly associated with the heavy pyrite constituent. Here the bowl reclassified selectively the relatively coarse pulp from the secondary circuit, overflowing the bulk of the quartz and only the finest sulphides for treatment, concentrating the sand product to five or six times the assay of the original ore and regrinding this high-grade material as fine as its assay warranted.

The net result has been a higher extraction, a coarser over-all grind and a reduced grinding cost.

The same principle has been applied successfully more recently in the dressing of copper sulphide ores for flotation. The sulphide mineral, being heavier than the gangue, concentrates readily in the bowl sand and is subjected to regrinding for more complete liberation of associated minerals. In this way the mill is not burdened with gangue material upon which no further work is required. Moreover, the finishing-bowl classifier protects the flotation operation against tramp oversize from preceding classifiers and assures a uniform product from the entire fine-grinding section, convenient for sampling and distribution to the succeeding units.

MEANS OF CLASSIFICATION

Cones, hydraulic classifiers and mechanical classifiers are the principal means of classifying pulp.

CONES

Classifying and dewatering cones once had a wide vogue and still are used in some pulp circuits. They were more or less troublesome because solids accumulated on the sloping sides, sloughing off periodically and plugging the discharge in the apex; the sand discharged was too dilute and contained too much slime; and sand was frequently carried over with the slime. Furthermore, large cones require considerable space and headroom.

It is of interest that the Homestake, long a user of cones for dewatering and classifying, selected mechanical bowl classifiers for their new mill addition.

In the Homestake's enlarged and rearranged plant are 10 cones of local make, 7 ft. in diameter and with 65-deg. sides. They dewater the pulp from the stamps and feed 10 rod mills which are in closed circuit with an equal number of Clark-Todd amalgamators and Dorr classifiers. Then the overflow passes to Dorr bowl classifiers and from them to ball mills in closed circuit with amalgamators and Dorr classifiers. Bowl classifiers make sand and slime products for leaching and filter pressing, respectively. Cones are used in the Dome plant, Ontario. There are two distributing cones and 16 classifying ones, $4\frac{1}{2}$ ft. deep, with

60-deg. sides and $\frac{3}{4}$ -in. spigot, which build up the circuit with sulphides from $3\frac{3}{4}$ to 9 per cent for selective grinding.

Cones of the Caldecott type, fitted with automatic discharge, are used in the four plants at Kolar, India, for dewatering and making tube-mill feed. In this cone the discharge continues to operate until there is a change in the pulp density. If the pulp in the discharge becomes too thin, the plug at the cone bottom closes, allowing the sand to accumulate. The density then increases to the desired point after which automatic control becomes effective.

Cone classifiers in Rand stamp mills consist of a nest of cones 4 ft. in diameter and 5 ft. deep. The overflow passes to secondary cones to be separated into sand and slime. The underflow is divided into as many portions as there are tube mills, each portion gravitating to a dewatering cone $5\frac{1}{2}$ by $7\frac{1}{2}$ ft., the underflow of which is the tube-mill feed. A little water is added to bring the moisture content to 30 per cent. The cone, however, is inefficient, according to T. K. Prentiss in *Bul., I.M. and M.*, April, 1935, in dewatering a finely ground pulp for tube milling. It has been found that the Dorr classifier is preferable for this purpose, and this machine is included in the flow-sheets of all of the latest plants.

HYDRAULIC CLASSIFIERS

Hydraulic classifiers range from simple V-shaped launders with a multiplicity of shallow settling pockets for the discharge of as many roughly sized products as desired to the more elaborate and efficient units of the Fahrenwald type, equipped with pockets having construction plates and jets of water for controlling the character of products collected in each pocket.

In general, hydraulic classifiers use more water than mechanical classifiers, the slime overflow is more dilute, the coarse portion or portions are at a much higher moisture content and the loss of head on the sands is considerable, which is objectionable in closed-circuit grinding. Also, these classifiers generally require a deslimed feed.

Hindered settling machines require no power except for the water used and are automatic in discharge, although they require a certain amount of attention. On many materials they give excellent results either in closeness of sizing or in concentration

of the denser particles. They are used, therefore, chiefly for two purposes—either in separating 6- to 200-mesh material into a series of closely graded portions for subsequent treatment or in concentrating the heavier mineral of a nonhomogeneous feed.

In practice, for preparing cyanide or flotation pulps, almost to the exclusion of all other types the mechanical, nonhydraulic classifiers are used because of their capacity, flexibility, continuous operation and uniformity of results. In addition, they require little attention and give a coarse portion or sand of low moisture content. Furthermore, the mechanical-type classifiers give a coarse portion that is as free from fines as any but the most efficient types of hydraulic classifiers. This is the result of the agitation and draining that take place in the course of the removal of the coarse portion.

MECHANICAL CLASSIFIERS

The impossibility of preparing clean, leachable sand and sand-free slime from an irregular feed of varying ore by double-cone classification led to the invention of the original Dorr classifier at the Lundberg, Dorr and Wilson mill at Terry, S. D. in 1904.

Mechanical dewaterers had been used before Johnson's shovel wheel, Seobey's belt dewaterer and other devices, including George Moore's cylinder with spiral conveyor. The Akins hollow spiral followed about 1910; and Philip Argall's double solid spiral, the Avoca, a few years later. The Seobey belt was developed as the Esperanza and used especially in Mexico; and more recently another cylinder with spirals is being introduced by Harding; while on the Rand a spiral draining sand from a cone has met some favor.

Akins Classifier.—The Akins classifier consists of a semi-cylindrical trough, usually set at a slope of from $2\frac{1}{2}$ to $3\frac{1}{2}$ in. per ft., in which rotates a helix or spiral at from 2 to 8 r.p.m. depending upon size and conditions to be met.

The feed is introduced through the side of the tank just below the pulp level. The heavier solids settle out and are carried upward out of the pulp by the spirals and discharged into a sand launder, while the fines flow in opposite directions to the overflow weir at the lower end of the tank.

The earlier types of machine used interrupted spirals and overflow lips below the shaft carrying the spirals at the lower end.

Two new types are now recommended by the manufacturers, the Colorado Iron Works Company, both of which have relatively deeper pulp depths with the overflow weir above the center of the shaft carrying the spirals at the lower end of the tank, and special construction prevents grit from working into the bearing. In the "high-weir" type recommended for coarser separations the spirals extend above the pulp level, and in the "submerged" type recommended for fine separations the spirals are below the pulp level in the lower end of the tank. Either type may be supplied with single or double spirals, and both are equipped with lifting devices.

Figure 31 shows a Duplex submerged-type machine.

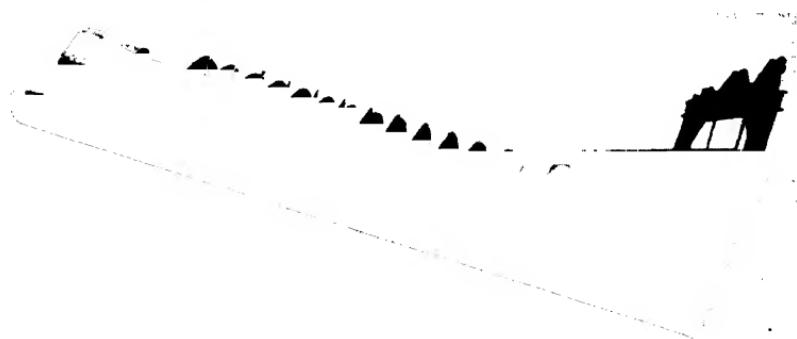


FIG. 31.—Akins classifier, Duplex submerged type.

The Dorr Classifier. Single-stage Type.—The Dorr classifier (Fig. 29) consists of a rectangular, inclined settling box of wood, steel or concrete, with the upper or discharge end open, in which are placed mechanically operated rakes or scrapers which carry the quick settling, coarse particles to the point of discharge at the open end. Each rake, of which there may be as many as six, is carried by two hangers, one at the discharge end and one near the overflow end. The rakes are raised, lowered and moved parallel to the sloping tank bottom by a suitable head motion. A lifting device is provided for raising the rakes several inches to clear the settling solids in case of a shutdown.

The feed enters continuously through a distributing launder near the overflow end. The more quickly settling particles fall to the bottom and are advanced up the inclined deck by the rakes and discharged. The agitation of the reciprocating rakes keeps

fine particles in suspension until they overflow the weir at the lower end.

The point at which a separation may be made is determined by the rake speed, overflow dilution and slope of tank bottom. The greater the rake speed the lower the dilution; and the steeper the slope the coarser is the separation; and conversely.

Bowl Type.—The Dorr bowl classifier (Fig. 32) is a combination of a shallow, circular bowl with a revolving raking mechanism

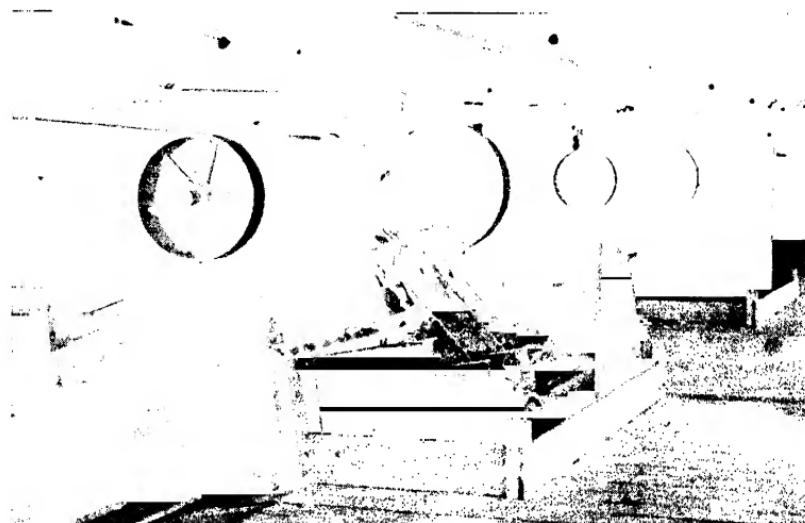


FIG. 32.—Rake discharge of Dorr bowl classifiers in Lake Shore plant, Kirkland Lake, Ontario.

superimposed on the lower or overflow end of a single-stage Dorr classifier. Feed enters through a loading well at the center of the bowl, and fine solids overflow across a peripheral weir. Coarse solids settle on the bowl bottom, are raked to a central opening and gravitate through it into the reciprocating-rake compartment.

It is used where a clean rake product is desired, where a separation at a fine mesh is to be made or where the overflow capacity must be large in comparison with raking capacity. It is essentially a two-stage classifier wherein the bowl oversize is reclassified in the reciprocating-rake compartment. The fines liberated in this recleaning step are carried back into the bowl and overflowed.

The critical size control is a recent improvement in the bowl classifier. It provides a positive and increased rate of circulation from the reciprocating-rake compartment beneath the bowl back into the bowl itself. It is of particular value in yielding a cleaner rake sand.

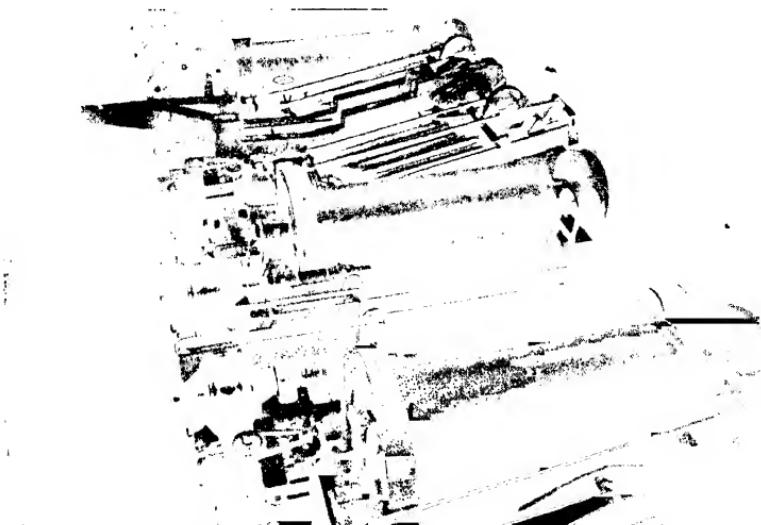


FIG. 33.—Grinding section at McIntyre Porcupine; note flotation cells between tube mills and classifiers for removing free gold.

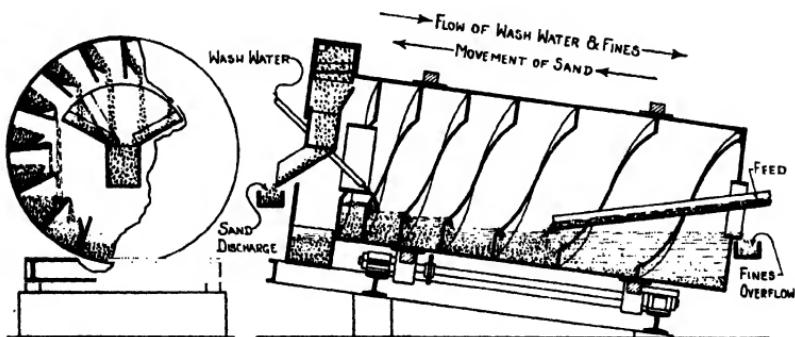


FIG. 34.—General arrangement of Hardinge countercurrent classifier.

Hardinge Classifier.—The Hardinge Company recently introduced a new type of mechanical classifier, as illustrated in Fig. 34. This classifier is a slowly rotating drum, on the inner surface of which is a screw flight attached to the drum, revolving with it.

The material to be classified is fed in at one end above the pulp level; and as the classifier rotates, the coarse particles that settle out are moved forward by the screw flight. The fines overflow through an opening at the feed end of the classifier. The sand or oversize is dewatered and elevated by buckets to the discharge hopper.

AGITATION

Agitation in cyanidation has a twofold purpose: First, to keep the solids in suspension while dissolution is taking place; and, second, to supply the required amount of oxygen. Violent agitation has no recognized value in itself, but ample oxygen supply is essential to promote the reaction between gold, silver and cyanide. Control of aeration during agitation is essential, since excessive aeration usually results in greater consumption of lime and sometimes of cyanide, with no increase in extraction.

Length of contact and dilution during agitation vary considerably for different ores. For gold ores agitation periods vary from 16 to 48 hr., probably averaging 30 hr., and for silver ores nearly twice this period. Dilution during agitation varies from 1 of solution to 1 of solids by weight to 3 of solution to 1 of solids, occasionally higher, probably averaging 2 to 1. Operation is continuous through a series of agitators, preferably three or more, to minimize any tendency for short-circuiting. Change of solution during agitation by means of a thickener placed between agitators often improves extraction, particularly when silver or other ores requiring long periods of agitation are being treated.

Three types of agitators are now in general use: Mechanical types (Devereux, Turbo, Wallace); air-lift types (Pachuca, Parral); air-mechanical type (Dorr).

The Dorr Agitator.—The Dorr agitator, in general use today, is a combination mechanical and air agitator designed for use in round, flat-bottomed tanks as large as 50 ft. in diameter and 25 ft. in depth.

Figure 35 shows diagrammatically, by arrows, the type of circulation maintained in the tank, whereby pulp and coarse solids from the bottom of the tank are elevated by means of air through the revolving hollow central shaft to a head casting above the pulp level and thence distributed through two revolving launders over the surface of the pulp. Coarse solids settling

on the bottom of the tank are raked to the central lift by two plow arms attached to the shaft.

The rate of circulation in the tank is controlled by means of the air supply to the central lift.

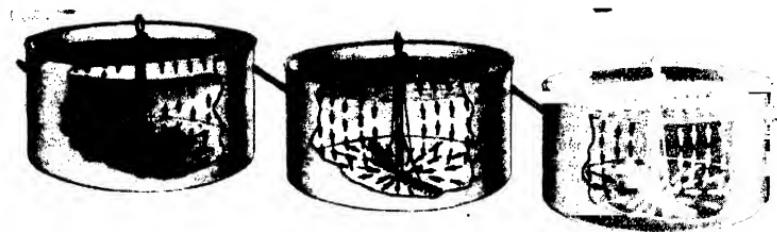


FIG. 35.--Diagram of direction of pulp movement, continuous agitation in series of three agitators.

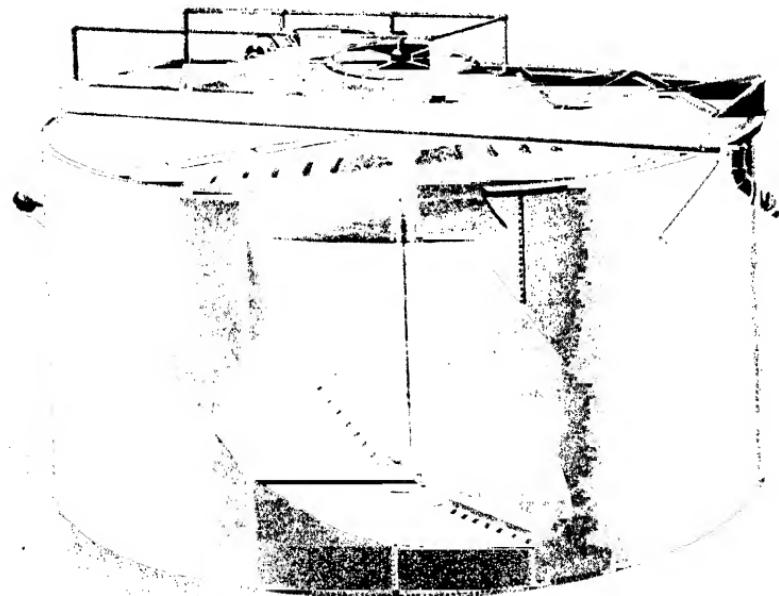


FIG. 36.--Phantom view of Type A Dorr agitator, recent type with I-beam supports.

Air is absorbed in the solution both from the air lift and from the series of pulp streams plunging into the surface of the pulp from the distributing launders.

Figure 36 shows the more recent type of Dorr agitator with I-beam supports, which requires minimum headroom and makes for ready accessibility.

The power required for both mechanical operation and air for agitators from 12 to 40 ft. in diameter will vary from about 1.0 to 0.5 hp. per 1000 cu. ft. of capacity.

Mechanical Agitators.—The first agitators used in the industry were of the simple rotating-paddle type. These, however, required a large amount of power and did not give uniform suspension or satisfactory means for controlling aeration in cyanidation.

Devereux.—The Devereux is a propeller-type agitator in which a propeller revolving centrally in a round, flat-bottomed tank at about 1-3 the depth of the tank above the bottom forces the pulp downward to the bottom where it is deflected along the bottom and up the tank sides, creating a vortex at the center in which air is drawn into the pulp. This machine, while simple mechanically, requires relatively high power. At one plant an 18- by 18-ft. Devereux requires 15 hp. when agitating a pulp containing 62 per cent solids.

Turbo and Wallace.—The Turbo and the Wallace are high-speed impeller types of agitator imparting high velocity to the pulp and are suited for relatively small-diameter flat-bottomed tanks. The Turbo may be provided with an impeller near the pulp surface whereby large amounts of free air may be drawn into the pulp and emulsified.

The Wallace consists essentially of an impeller or rotor near the tank bottom and at the lower end of a stationary vertical pipe which extends nearly to the pulp surface. The action of the impeller creates a circulation of pulp down the pipe and into the impeller. When the pipe is properly placed, free air is drawn into the top of the pipe together with pulp and disseminated throughout the tank. The pipe may have auxiliary openings to draw in pulp from different levels in the tank.

Both of these agitators have been used in combination with Dorr agitators for supplying large quantities of air to the pulp when treating ores or concentrate requiring an unusual degree of aeration.

Pachuca.—The Brown or Pachuca agitator, as it is most commonly known, originated in New Zealand in the early days of

cyanidation, the invention of F. C. Brown. It later became popular in Mexico, and to a limited extent its use has spread to other countries.

The Pachuca agitator is an all-air agitator which consists of a central stationary air-lift pipe in a circular cone-bottomed tank of varying dimensions, usually from 5 to 18 ft. in diameter and 20 to 60 ft. in depth.

Relatively large quantities of high-pressure air are required, and with some ores considerable difficulty results from the building up and sloughing off of solid masses from the tank sides. The height of the Pachuca is a further disadvantage and usually necessitates pumping of the pulp.

PACHUCA AGITATORS
(Air-consumption data from practice)

Size, feet	Cubic feet air per minute	Air pressure	Dilution	Cubic feet air per 1000 cu. ft. tank
15 by 45	65	0.6:1	7.0
15 by 60	122	42.5	1.2:1	11.0
18 by 22	79	38	1.4:1	11.6
15 by 40	70	14	9.0

CHAPTER VII

SAND TREATMENT

Ores may be separated into sand and slime, and the sand leached by upward or downward as well as upward and downward percolation of cyanide solutions. Leaching practice has decreased in recent years with the development of closed-circuit grinding and now is seldom incorporated in new plants.

Ores are known which need only to be crushed coarsely to $\frac{1}{2}$, in., and the whole leached. Deposits of ore amenable to coarse crushing, however, are rare. The gold must be on cleavage planes or in the cavities left by previous natural leaching of the ore in place.

Sand leaching is carried out in vats ranging in capacity from 30 to 900 tons. Sand for leaching is separated from slime in cones, V-boxes, classifiers, and in collecting vats filled by distributors—the overflow in each case being slime. As a rule, leaching is a simple process, the objective being a vat of well-mixed neutralized sand, ample contact with strong and weak cyanide solutions, water washes, aeration of the sand and solution and rapid filling and discharging. It is a cheap and effective process for clean ores when fine grinding is not necessary to good extraction.

Classification for Leaching.—The importance of classification prior to leaching cannot be overemphasized. Sands that are essentially free from colloidal material behave quite differently from the same type of sand with a small percentage of colloid. The two following examples from plant practice are illustrative.

The *Golden Cycle* mill at Colorado Springs, Colo., grind roasted siliceous ore in Chilean mills through 20 mesh before sand-slime separation at about 200 mesh. Prior to the development of the Dorr bowl classifier at this plant, this separation was made in Dorr classifiers which produced a sand containing about 15 per cent minus 200 mesh, with an appreciable amount of contained colloid. The leaching rate in the sand tanks varied from 0.7 to 1.4 in. per hr.

Following the adoption of the bowl classifiers, which produced practically colloid-free sand, the leaching rate increased to about

10 in. per hr., and leaching results were much more uniform. The following is a typical screen analysis of the bowl-classifier sand when the overflow contained 2.6 per cent plus 200 mesh.

+30 mesh	- 6.7 cumulative
+60 mesh	- 53.2 cumulative
+100 mesh	- 73.2 cumulative
+150 mesh	- 84.2 cumulative
+200 mesh	- 96.0 cumulative
-200 mesh	- 100.0 cumulative

The *Homestake Mining Company* grind their ores in closed circuit with rod and ball mills through about 80 mesh and, following

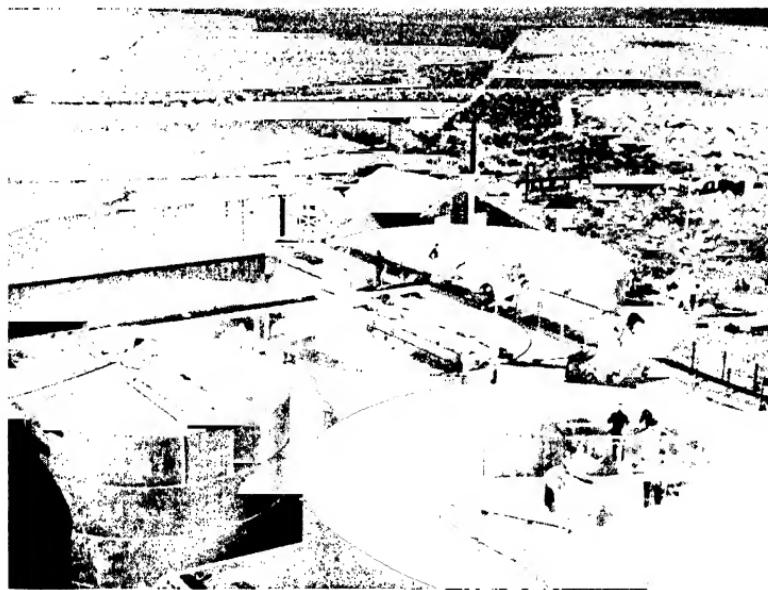


FIG. 37.—Golden Queen Mining Company, Mojave, California; showing Type A Dorr agitators (filled) and Dorr centre-pier thickeners (partially filled).

amalgamation, make a sand-slime separation at about 2 per cent plus 200 mesh. In their new plant, Dorr bowl classifiers make this separation prior to leaching and slime treatment. The Homestake ores contain an appreciable amount of pyrite which is gold bearing, and the bowl classifiers are adjusted to produce a maximum of -200 mesh granular material in the sands, including very fine pyrite, but to remove all colloidal material. Even with

this unusually fine sand, excellent and uniform leaching results are obtained at rates up to $2\frac{1}{2}$ in. per hr. in 11-ft. depth tanks. The following is a typical screen analysis of this sand:

TYPICAL SIZING OF SAND
(Cyanide 3 sand plant)

Mesh	Per cent	Per cent cumulative
+50	0.0	0.0
+80	2.4	2.4
+100	5.1	7.5
+150	16.9	24.4
+200	28.3	52.7
-200	47.3	100.0

Fresnillo Leaching.—At Fresnillo, Mexico, according to W. E. Crawford, cited later, the pulp is separated into sand and slime.

The sand plant, during normal operations, handles 17 per cent of the total tonnage on the charge system, in five 60 ft. diameter by 16 ft. deep steel tanks, each having a capacity of 1815 tons of dry material. The pulp is delivered to the sump ahead of the tanks by a launder from the sand-slime separation plant. A 5-in. Krogh sand pump discharges the pulp from the sump into a diversion box connected to individual Butters distributors by a series of launders, arranged so that the sand can be diverted to any tank. The time of filling is 3 to 6 days, depending on the amount of sand produced in the mill.

When the tank is full it is allowed to drain, and a strong bath containing 0.25 per cent KCN and 0.05 per cent CaO is added. This wash is normally continued for 5 days. After the strong solution has been drained off, the sand is given a bath of weak solution containing 0.075 per cent KCN and 0.05 per cent CaO for a period of 5 days. Finally, a water wash of 660 tons is added.

After draining under a low vacuum the tank is discharged onto conveying belts through chutes; contract labor is used to shovel the sand which will drop through the chutes by gravity. At the end of the conveyor belts the sand is mixed with the slime discharged from the filter plant; the combined tailings flow by

gravity to the tailings pond, where they are impounded. The sand has 14 per cent moisture when discharged from the tank, this low moisture helping to conserve water.

Homestake Leaching.—Although it is finely ground in water, *Homestake sand can be cyanided by gravity leaching to yield more than 80 per cent of the gold. Sieve analyses and assays reveal that the finer the sand (minus 200 mesh) the higher the extraction (90 per cent), compared with 54 per cent of the plus 50 mesh, which is a small part of the total.*

Leaching is done in two sand plants, one with four vats 55 ft. in diameter by 13 ft. deep, the other with 19 vats 44 ft. in diameter by 10½ ft. deep. They are filled through Butters-Mein distributors. Lime is added to the pulp as it leaves the final set of classifying cones. The filter bottom consists of the usual wooden frame covered with coco matting and 10-oz. canvas duck. A sieve analysis of a charge, cumulative on each mesh, is as follows: 7 per cent on 100 mesh, 24 per cent on 150 mesh, 53 per cent on 200 mesh, with 47 per cent passing 200 mesh.

Repeated aeration is essential to maintain enough oxygen in the solutions to effect extraction, according to A. J. Clark in *E. and M.J.*, Oct. 12, 1931. This is done by introducing low-pressure air under the filter bottom of the leaching vat.

The leaching practice is designed to effect a progressive enrichment of the solution before precipitation, the effluent from the latter part of a treatment being strengthened but not precipitated before being returned to the top of another charge. Solution and zinc are conserved by this system. The total solution used amounts to only ½ ton per ton of sand. The water draining from the newly filled charge contains sulphates and thiosulphates. After the second aeration, when cyanide solution has been added to the top, these compounds appear in larger quantity. Later, they are followed by thiocyanates, which in turn are followed by the first traces of cyanides. Gold appears in the effluent soon after the first trace of cyanide is noted. Thiocyanates, the main source of cyanide loss, are stable compounds, but although they build up in solutions, they do not affect extraction.

An extremely low alkalinity is favored. A cyanide of 49 per cent NaCN equivalent is used. Strong solutions carry 0.08 to 0.09 per cent, and the effluent 0.035 per cent. Cyanide consumption is ¼ lb. per ton treated.

A typical sand-leaching cycle follows: The charge of 654 tons assayed \$2.51 gold per ton and after treatment 45 cts., giving an extraction of 82.1 per cent.

SAND-TREATMENT CYCLE AT THE HOMESTAKE

Operation	Period		Remarks
	Hours	Minutes	
1. Filling.....	8	20	Pulp containing 43 per cent solids joins discharge of lime mill and enters water-filled vat through Butters-Mein distributor. Overflow, after clarification, is reused in classifiers
2. First draining.....	15	40	Effluent wasted
3. First aerating.....	14	..	Gage pressure, 6½ lb.; bottom solution valves closed
4. Strong-solution leaching....	16	..	Rate, 1½ in. per hr.; solution strength, 0.09 % NaCN; gold in solution, \$1.94 per ton. Displaced moisture wasted
5. Second draining.....	12	..	Effluent wasted
6. Second aerating.....	11	..	Gage pressure, 7 lb.; bottom solution valves closed
7. Strong-solution leaching....	14	..	Rate, 1½ in. per hr.; solution strength, 0.09 % NaCN; gold in solution, \$2.30 per ton. Effluent to low-solution sump, followed by precipitation and wasting of barren solution
8. Third draining.....	10	..	Effluent to weak-solution sump, followed by precipitation; barren solution to storage for reuse
9. Third aerating.....	10	..	Gage pressure, 7 lb.; bottom solution valves closed
10. Weak-solution leaching....	12	..	Solution strength, 0.055 per cent NaCN; gold in solution, 1 cent per ton. Effluent to weak-solution sump, followed by precipitation, and discharge of barren solution to storage for reuse
11. Washing.....	29	30	Rate, 1¾ in. per hour. Effluent during first 6 hr. to weak-solution sump, followed by precipitation and discharge of barren solution to storage for reuse. Effluent during next 21 hr. to strong-solution sump for reuse. Effluent during last 2½ hr. to low-solution sump, followed by precipitation and discharge of barren solution to waste. Wash water from cone overflow
12. Sluicing.....	1	30	
Total.....	154 hr. or 6 4 days		

Golden Cycle Leaching.—At the Golden Cycle, Dorr bowl classifiers with 15-ft. diameter bowls and 6-ft. wide rakes make

a sand-slime separation as shown in the accompanying table. Classifier feed amounts to 1200 tons per day, and about 1200 tons of solution backwash are used on the sand deck.



FIG. 38.—Interior, Homestake Mining Company, Lead, South Dakota, showing 20-in. diameter *Lorr* down classifiers.

SAND-SLIME SEPARATION AT GOLDEN CYCLE

Tyler standard size	Percentage		
	Feed	Sand	Slime
On 14-mesh screen.....	0.2 cum.	0.2 cum.	
On 16-mesh screen.....	0.9	0.8	
On 20-mesh screen.....	5.7	7.4	
On 30-mesh screen.....	19.8	33.7	
On 40-mesh screen.....	30.4	52.8	
On 60-mesh screen.....	53.2	85.3	0.8 cum.
On 100-mesh screen.....	56.7	92.7	3.5
On 150-mesh screen.....	63.3	98.0	15.9
On 200-mesh screen.....	69.3	99.4	29.1
Through 200-mesh screen.....	100.0	100.0	100.0
Moisture.....	86.7	23.3	93.8

The sands are conveyed to 10 leaching vats 50 ft. by 9 ft. 9 in. deep with a capacity of 900 tons each. The initial leaching

period is 48 hr. with solution containing 2 lb. cyanide per ton. This solution goes to the precipitation presses. Sand charges are drained and aerated at least four times for 8-hr. periods during the 6- to 8-day treatment cycle. Barren-solution washes and a water wash follow. Two men using $2\frac{1}{2}$ -in. hose and using water at 120-lb. pressure discharge the vats in $3\frac{1}{2}$ hr.

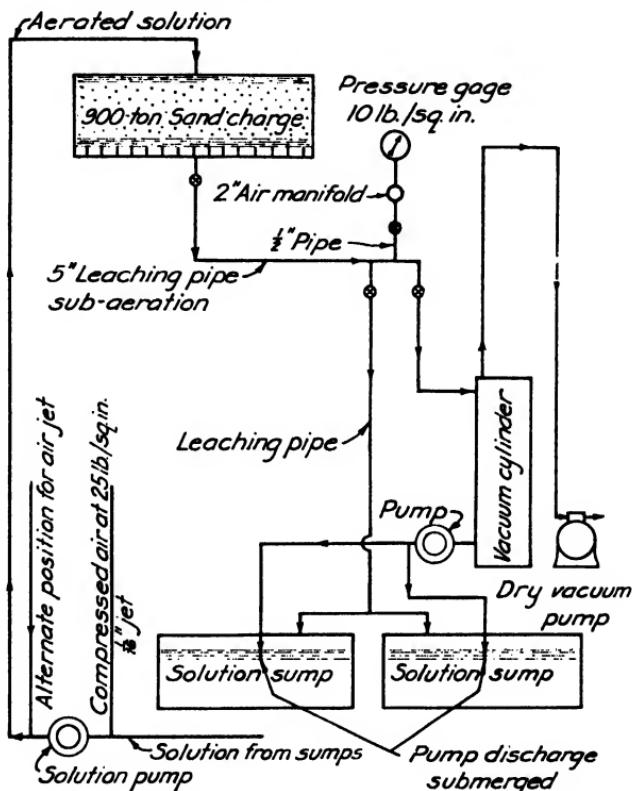


FIG. 39.—Aeration of solution and subaeration and vacuum drainage of sand charge at Government Gold Mining Areas Plant, Transvaal.

Rand Leaching.—In *Bul., I.M. and M.*, April, 1935, T. K. Prentice tells of practice at the plants of the Central Mining-Rand Mines group which milled 14,700,000 tons of ore during 1933. The percentage of ore treated as sand is 47. It averages 2.884 dwt. gold per ton, and 5 per cent is coarser than 48 mesh, 46 per cent finer than 48 and coarser than 100 mesh, 35 per cent finer than 100 and coarser than 200 mesh and 13 per cent finer

than 200 mesh. A clean sand is made in 8- by 8-ft. cones the underflow of which is fed to sand collectors 40 to 55 ft. in diameter and holding 400 to 850 tons of sand each. These tanks are filled by distributors or by lengths of 6-in. hose. The over-flowing slime from cones, classifiers and sand collectors gravitates to slime collectors. The sand collected weighs 23 to 21 cu. ft. per dry ton and after draining contains 12 per cent moisture. Unless cyanidation is done in the collector, the sand is transferred to a treatment tank immediately below the collector or on the same level adjacent to it. Treatment lasts 4 to 7 days with 1.7 tons of solution per ton of sand, the strongest solution being 0.12 per cent cyanide. Solutions of medium strength and of low strength follow. Where treatment tanks are superimposed, half of the cycle is done in each level. Additional aeration is thus obtained. The system for aeration of solution and subaeration and vacuum drainage of a sand charge on the Rand, shown in Fig. 39, is described by F. Wartenweiler in *Trans. 112, A.I.M.E.*, 1934.

LEACHING COARSE ORE

Metallurgists probably remember the simple and satisfactory treatment of \$2 ore of the Wasp No. 2, South Dakota, and of \$4 ore of the Borneo Company in Sarawak. Crushed to about 4 mesh, these ores were cyanided at low cost and fair extraction. Now the Mountain Copper Company, at Matheson, Shasta County, Calif., is crushing, screening and leaching 700 tons a day of gold-bearing gossan.

Mountain Copper Company.—The results of careful sampling and tests made in a 4-ton plant proved the ore to average \$1.85 per ton and that an extraction of 75 to 87 per cent could be obtained by simple leaching. No free gold has been detected. The gossan consists of limonite and quartz and is porous and friable. It also contains 0.4 per cent copper, a little arsenic and mercury, 50 to 55 per cent iron and 5 to 10 per cent silica. The material weighs 104 lb. per cu. ft. It is mined by electric shovel, hauled in 8-ton motor trucks to a rail grizzly, crushed by a Buehanan jaw breaker and Anaconda type rolls and screened, producing 3 $\frac{1}{2}$ -in. mesh material. According to G. J. Young in *E. and M.J.*, June 22, 1931, leaching is done in 10 wooden vats 25 ft. 8 in. diameter by 12 ft. deep, holding 276 tons each. The vats are filled by belt conveyors and trippers to within 3 in. of

the top, and settlement during treatment is 6 to 9 in. Solution inflow is started from the bottom upward when a vat is half full, the strength being 0.8 lb., NaCN equivalent, "aero" cyanide, being used. All subsequent washes are from the top. The treatment cycle follows:

LEACHING CYCLE FOR GOSSAN

Solution	Strength, pounds NaCN	Quantity, tons	Period, hours
Strong.....	0.8	195	16
Weak.....	0.5	90	24
Barren.....	0.4	180	20
Water.....	...	75	7½
Total.....	...	540	67½

Gold is precipitated on zinc dust, and the reaction is enhanced by the presence of dissolved mercury. Cleanup and melting follow standard practice, but the bullion is only 455 fine in gold and 180 in silver. The indicated extraction from \$1.85 gossan was 73 per cent at a cost of 43 cents per ton treated. Chemicals consumed were 0.344 lb. cyanide, 8.35 lb. lime and 0.0257 lb. zinc. The quarry equipment cost \$25,000, and the cyanide plant \$75,000. Thirty-one men are employed.

Wasp No. 2.—The ore treated was a quartzite with the gold in the cleavage planes. It was crushed to 6 mesh by a Gates gyratory and Gates rolls, with screens in circuit. The conveyor-filled wooden leaching vats, with central discharge gate, held 55 to 100 tons and treated 105 tons daily. Charging occupied 2 to 2½ hr., and discharging by sluicing 2½ to 3 hr. The leaching cycle was as follows:

LEACHING CYCLE FOR QUARTZITE

Solution	Strength, pounds KCN	Quantity, tons	Period, hours
Strong.....	6	15	3 to 4 (standing)
Weak.....	2½ to 3	40 to 50	
Water.....	6 to 9	
Total.....	70 to 75	96 to 100

According to C. H. Fulton in *The Cyanide Process in the Black Hills of South Dakota*, 1902, actual extraction was 80 to 85 per cent at a cost of 83 cts. per ton. Cyanide consumption was 0.63 lb. per ton ore, lime $2\frac{1}{2}$ lb. per ton solution and zinc shavings 0.9 lb. per oz. bullion.

LEACHING OF CLAY ORE

Bidi, Sarawak, Borneo.—To solve the problem of treating a clay ore in which the gold was finely disseminated, the author was able to suggest a unique method which he had seen described twenty-five years previously in a paper, "Occurrence and Treatment of Gold Ore at Bidi, Sarawak, Borneo," by T. C. Scrutton, which appeared in *Trans. 15, I.M.M.*, 1905–1906, in which clay ore was treated by leaching in relatively large lumps. Its successful application in this instance is our justification for including a detailed description of the method in the present volume.

The auriferous deposits at Bidi consist of a series of unconnected hills lying upon the weathered surface of limestone. These hills consist of clays and earth containing boulders of stone or varying structure—pure silica, silicified sandstones and shales. The clays and earths carry from 2 to 15 dwt. gold per ton, averaging 5 dwt.; the stone, from 3 to 30 dwt. per ton, averaging 8 dwt. The proportion of stone to clay occurring in the deposits is roughly 1 to 5.

The ore is trammed to the six ore bins. The clay ore bins are four in number, and their arrangement constitutes an important feature in the direct treatment of clays.

To quote Scrutton:

To obtain rapid and easy leaching of the clayey material, it is necessary

1. To keep the fine earthy and sandy material separate as far as possible from the more plastic clayey material and to treat the two separately in different vats.
2. To ensure that all earthy and clayey material, when charged into the vats, is in the form of balls firm enough to maintain their form when charged into the vat and of a size varying from 3 in. in the case of the clayey material to $\frac{1}{4}$ in. in the case of the finer.

The first of these conditions is attained by providing four separate clay bins and carefully selecting the material from each according to its

tendency to break up and form fines or to agglomerate and form balls.

The second is provided for by running the clay from the tippler down to a fan-shaped chute, about 40 ft. long and inclined at 60 deg. into the ore bin; thus the masses of clay are broken up and formed into balls by rolling down the chute. The wetter and more plastic the clay treated the longer and steeper must this chute be to ensure the clay's being in a leachable condition on arriving at the ore bin. The clay ore bins are rectangular and discharge through bottom sliding doors into wagons running below.

Vats.—The leaching vats are 30 in number, 18 of 100 and 12 of 50 tons' capacity; the former are 6 ft., and the latter 3 ft. deep, all of 27 ft. 6 in. diameter. They are constructed of $\frac{3}{8}$ -in. mild steel plates, riveted with $\frac{3}{2}$ -in. rivets, $1\frac{3}{4}$ -in. pitch.

Charging.—Side-tip wagons are employed and are filled direct from the stone and clay ore bins, whence they are run direct over the vats and tipped; two pairs of rails run over each vat, arranged at such a distance apart as to require a minimum amount of shoveling to level off the material in the vat.

The following are the principles regulating the method of charging:

1. Clay must be charged in layers not more than 3 ft. deep; if this depth be exceeded, difficulties are experienced in leaching and washing, resulting in impaired extraction.
2. Coarse material must be kept separate from fine to ensure good leaching.
3. In the event of charging two different classes of stone and clay into the same vat, the operation must be conducted so as to leave the material as far as possible in horizontal, not vertical, layers.

In the case of the 50-ton vats, which are only 3 ft. deep, they are filled with clay from one ore bin, no further precautions being necessary.

With the 100-ton vats, in order to treat as large a quantity of clay as possible, sufficient clay is charged into the bottom of the vat to form a layer 2 ft. deep; this is then leveled off, and sufficient stone charged to form a layer $1\frac{1}{2}$ ft. deep; this having been leveled off, the vat is filled to the top with clay, giving an upper layer of 2 ft.; thus the layer of stone, by forming a porous bed in the middle of the clay, prevents the formation of channels throughout the whole mass and, by separating the clay into two thin layers, renders leaching comparatively easy; by charging in this manner equally good extractions are obtained from the clay treated in the large as in the small vats.

To ensure obtaining the correct tonnage, it is necessary to fill the vats until the clay stands about 2 in. above the top of the vat, as the material when properly charged lies exceedingly loosely and, after

solution has been on for a short time, sinks down to 9 in. below the top of the vat.

Discharging.--This is effected in the usual manner by shoveling the material through four bottom-discharge doors into wagons running on two lines of rails below the vats.

Treatment.--The usual treatment for 100-ton vats occupies 10 days and gives the following results:

Charge.....	5.4 dwt.
Discharge.....	1.2 dwt.
Extraction.....	78%
KCy consumption.....	0.97 lb. per ton

The solutions used are:

Strong solution, containing.....	0.07% KCy
Sump solution, containing.....	0.05% KCy

These solutions have to be kept at a definite standard of alkalinity, which is effected partly by putting a certain quantity of lime into each wagon at the bottom of the main incline and partly by the addition of soda to the sumps; the quantities of each used are regulated by the working of the zinc boxes. The average consumption of lime is 8.8 lb. per ton, and the maximum amount of soda usually added 0.14 lb. per ton of solution.

So long as this standard of alkalinity is carefully maintained, the solutions give very little trouble. If, however, the alkalinity gets too high, the solutions become dirty, foul smelling and full of arsenic and antimony; if too low, the consumption of cyanide is so great as to show no precipitate with AgNO_3 after once running through a vat, extraction, of course, suffering accordingly.

The first filling of strong solution is put on by very slow upward leaching, so as to disturb as little as possible the fine material lying loosely in the interstices between the larger balls of clay. The vat is gradually filled by this means, the operation, if properly conducted, occupying about 3 hr.; as the charge in the vat becomes soaked in solution it settles down, finally sinking to about 9 in. below the top. About 30 tons of solution is required for the first filling, but of this only about 20 tons can be drained off, the remainder being absorbed by the clay.

The vat when full is shut up and allowed to soak for 4 hr.; it is then opened and allowed to drain at such a pace as to ensure its being just dry in another 4 hr., when it is pumped up again, the solution being run on to the top of the charge, and the vat leached by gravity in the usual manner. This alternate filling by downward leaching, soaking and draining is carried on until about 150 tons of strong solution has been put on the vat, i.e., for about four days.

Sump solution is then substituted, the first filling being put on by upward leaching, and the subsequent filling by downward leaching, in the usual manner. The effect of using slow upward leaching in the middle of the treatment is to lighten the charge and form new channels for solution by altering the direction of pressure.

The downward leaching is continued as before until about 180 tons of sump solution has been given. This usually occupies about 5 days; the actual time of treatment, however, is judged by the amount of gold extracted according to the solution assays. When the solutions coming away contain only $\frac{3}{2}$ dwt. per ton of gold, a final water wash is given, and the vat discharged.

In order to ensure a regular flow of solution through the zinc boxes, it is necessary to divide the vats into two lots, one lot being drained while the other is soaking.

After repeated trials the foregoing method has been found to give the best results for clayey material charged direct into the vats. Given that the material has been properly charged into the vat, solution of the gold takes place almost immediately, on contact with the fresh cyanide solution, the metal being in an extremely fine state of subdivision, and lumps of clayey material containing only about 16 per cent moisture being readily permeable by solution.

Practically, the total gold contents of the clays are dissolved by the cyanide; this was shown by taking a number of samples of the material discharged from the vats and applying repeated washings of water; by this means alone a final extraction of over 95 per cent was obtained.

The problem, then, in order to ensure good extraction, is to wash out the auriferous cyanide solution which has been absorbed by the balls of clay. This cannot be effected by direct washing, in the ordinary sense of the term, the lumps of clay being only very slightly pervious, and the interstices forming easy channels for solutions; but it must be brought about by diffusion between the solutions rich in gold remaining in the dissolved clay, and the solutions containing practically no gold being pumped into the vats.

In order to obtain the best results, the point to be aimed at is to give the charge as much fresh solution as possible, consistently with sufficient time of soaking to allow of a certain amount of diffusion between the fresh solution and the auriferous solution remaining in the clay; prolonged soaking has not been found to give good results, doubtless owing to the slower diffusion of liquids carrying nearly the same quantities of gold in solutions, as compared with those differing widely in gold values. There is, however, a decided limit to the amount of solution which can be used, as it has been found that unless a sufficient time of soaking be given to allow the new solution to permeate the whole of the charge, the extraction suffers considerably, the new solutions coming away by

certain easily formed channels and absorbing very little of the richer solutions contained in less readily permeable parts of the vat.

Likewise slow draining off, *i.e.*, at the rate of 5 tons solution per hour, is a necessity. If this rate be exceeded, a much larger proportion of moisture remains in the vat, doubtless on account of the solution's descending too quickly into the vat to allow the small particles of solution lying in the interstices to agglomerate and descend with the mass of liquid; they are accordingly cut off by the air and left.

Extractor House.—The precipitation is effected in the usual manner by zinc shavings, the zinc boxes being 16 in number and containing in all 936 cu. ft. zinc. At the rate of flow given this allows 1 cu. ft. zinc for 1 ton solution per 24 hr.

Considering the large amounts of antimony and arsenic contained in the charge, the precipitation gives very little trouble, though from time to time the solutions become abnormally foul, and the precipitation is impaired.

When working satisfactorily, 0.5 per cent cyanide solution entering at 2 dwt. per ton is reduced to 3 gr.

CHAPTER VIII

SLIME TREATMENT

Slime treatment, as commonly used by metallurgical engineers, includes thickening, agitation and filtering and as applied to cyanidation also includes washing by countercurrent decantation (C.C.D.) or filters. "Slime" is the general term used to describe the finer portion of pulp in a combination sand and slime-treatment plant and is usually finer than 100 mesh. The so-called "all-slime" type of plant is that in which all of the ore is ground through a relatively fine mesh such as 100 mesh and where no separate treatment of sand and slime is provided.

Equipment used and methods employed are discussed, with examples from practice.

THICKENING

General.—Thickening or dewatering may be defined as the removal of a portion of the liquid from a pulp or slime made up of a mixture of finely divided solids and liquids.

The early methods of thickening employed plain, flat-bottomed tanks into which the pulp was fed until the tank was full. The solids were then allowed to settle as long as required, the top liquid was decanted, the settled solids were discharged and the operation was repeated. Such settling was usually carried out in a number of tanks so that a regular cycle of filling, settling and discharging could be maintained. Later it was found that feeding the tank behind a baffle allowed some decantation of clear liquid while still feeding, and this also was introduced.

Attempts to make thickening continuous, by using hopper-bottom tanks or cones, were not entirely successful. In these tanks the feed ran in continuously, settled solids were drawn off through a spigot and solution overflowed continuously at the top of the tank. The chief drawbacks to this method were the multiplicity of units required and the fact that settled solids hung up on the sloping sides which made it extremely difficult to obtain anything approaching a discharge of uniform density.

Dorr Thickener.—The invention of the Dorr thickener made possible the continuous dewatering of a dilute pulp whereby a

regular discharge of a thick pulp of uniform density took place concurrently with an overflow of clarified solution.

Scraper blades or rakes, driven by a suitable mechanism, rotating slowly over the bottom of the tank, which usually slopes gently toward the center, move the material settled on the bottom to a central opening or discharge. The rakes revolve at a speed sufficient to move the material as fast as it settles without enough agitation to interfere with settlement.

Dorr thickeners are used in the metallurgical field to thicken prior to agitation and filtration, in the countercurrent washing of cyanide slime, for thickening ahead of flotation, for thickening concentrates and for dewatering tailing to recover the water for reuse in the mill.

The standard construction of Dorr thickener mechanisms is of iron and steel. The tanks are usually made of steel or wood for medium sized machines, but in the larger sizes they are often constructed of concrete or earth or a combination of these materials. Various types of Dorr thickeners, which it is unnecessary to describe here, are available for specific uses, including constructions to resist corrosion. Power requirements are low, about $\frac{1}{2}$ hp. for thickeners less than 50 ft. in diameter. Attendance and repairs are also low.

Figures 41 and 42 illustrate the development of the original central-shaft type of thickener from one using superstructure to the present beam type of support whereby headroom is saved and an improved mechanism running in oil is made conveniently accessible to the operator.

The *Dorr tray thickener* (Fig. 42) has been developed to meet the definite demand for large settling area in limited space. Each compartment increases the capacity approximately to the same degree as an additional thickener unit, of the same size, without increasing the floor space required. The tanks are divided into two or more settling compartments by means of steel trays or diaphragms suspended from the sides. The mechanism is made up of a central vertical shaft, driven by worm gear and with radial arms attached above each tray. These arms carry plow blades set at an angle, and as the mechanism slowly revolves they move the settled material to a discharge opening at or near center.

Several types of tray thickeners have been developed, classified according to the arrangement of feed and discharge in the various

compartments. Probably the most generally applicable is the balanced-type tray thickener. In this each compartment has a separate feed and overflow, but the settled solids from all compartments are brought together and are discharged through a central outlet from the bottom compartment.

The *Dorr traction thickener* is the type most frequently used for heavy-duty work with large tonnages. The machine is so called because the thickening mechanism is driven by means of a tra-



FIG. 40.—Dorr traction thickener.

tion wheel which travels around the periphery of the tank on a rail. The mechanism, which is slowly carried around the tank as the traction wheel travels, consists of a truss to which are attached the raking blades which sweep the floor of the tank. This truss is supported at the center of the tank by a column. Power is brought in through a conduit which comes up through this column and is carried out along the truss to the drive unit which operates the traction wheel (see Fig. 40).

The *Dorr Torq Thickener* is a new development in the unit thickener field and is illustrated in the photograph of an 8-ft. diameter model in Fig. 41. The raking mechanism is supported from a stationary central pier on which the driving motor and gear reduction are mounted. The feed may be introduced at the center through a launder suspended above the top

of the liquid level or by means of a siphon feed through the hollow central pier. In addition to compact strength provided with the central pier construction, the machine is characterized by a new rake-lifting feature whereby when overloaded the rakes raise backward and upward at an angle to clear the overload and at the same time maintain a full raking load until the overload is removed and the rakes assume their normal position. Figure 41 shows the model rakes in a raised position such as would be assumed under very heavy overload in practice.

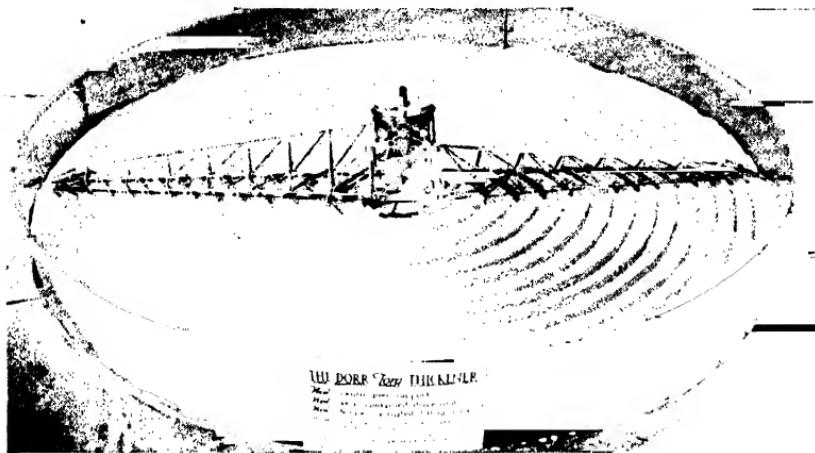


FIG. 41.—Model of Dorr Torq thickener.

The *Dorr washing thickener* is of the multiple-tray type and is, in effect, a complete countercurrent decantation plant in a single compact unit. It is adapted to the needs of the small chemical and metallurgical plants where a relatively small tonnage of finely divided solids must be washed free from a solution in a minimum space. Four to five stages of washing are usually provided in separate tray compartments.

Multiple-stage washing is effected by operating the tray compartments in series rather than in parallel. Feed enters, and strong solution leaves the uppermost compartment. Washed solids are discharged from the bottom compartment. Solids, after each successive settlement, gravitate from one compartment to another through sludge seals in each tray. Wash water, introduced in the bottom compartment, rises successively through the compartments in a direction countercurrent to that of the

sludge. One tall thickener thus does the work of several relatively shallow thickeners of the same diameter. Floor space is

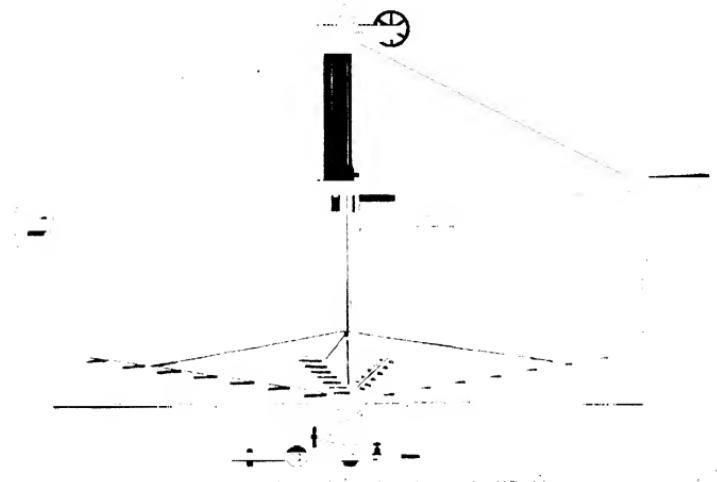


FIG. 42.—Original Dorr thickener, installed at Mogul Mining Company, Pluma, South Dakota, 1906.

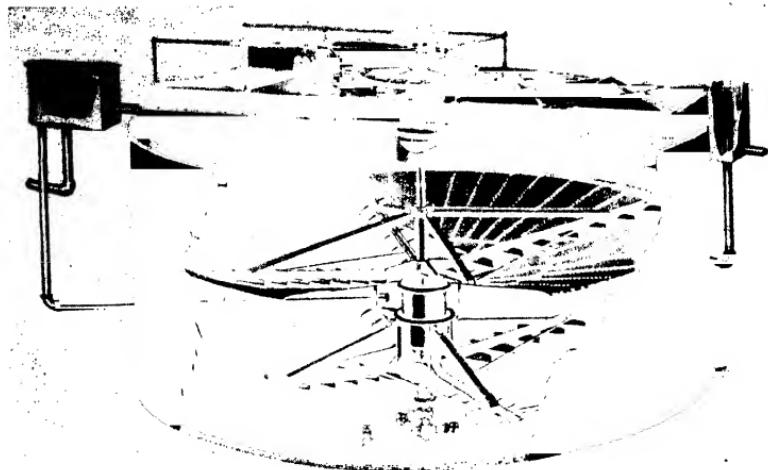


FIG. 43.—Type ATB, Dorr balanced-tray type thickener.

conserved, heat insulation is simple and effective and power is reduced. Furthermore, all control of the washing operation is centered at a single point.

Diaphragm Pump.—The diaphragm pump is essentially one for low lifts and is particularly adapted to handle metallurgical pulps economically and with minimum attention. The Dorco diaphragm pump was developed primarily as a means of controlling the density of the underflow from the Dorr thickener. It has proved the most satisfactory method of accomplishing this because of its positive, uniform displacement, which can be regulated at any rate of flow within the limits of its capacity. When the underflow from the thickener is too thin, a simple regulation of the pump will decrease the rate of discharge until its density is correct. Once the pump has been regulated it is seldom necessary to change the adjustment unless there is a decided change in the feed to the thickener. Cord diaphragms constructed on the same principle as cord tires give appreciably longer life than the original fabric construction. Diaphragm pumps of different types are available from several manufacturers.

Genter Thickener.—The Genter thickener is a filter thickener wherein the solids are removed from suspension by filtration and the clear liquor is passed through a filter medium which retains the suspended solids on its exterior. The solids, in cake form, then drop periodically and automatically to the bottom of the thickener tank and are moved to a central discharge point by means of a slowly revolving rake mechanism (see Fig. 44).

The filter area is subdivided into several cylindrical filter elements placed in groups in supporting frames which are assembled around a filtrate-collecting receiver with automatic valve centrally placed in a circular tank up to $18\frac{1}{2}$ ft. in diameter. The entire tank is filled with pulp, thus keeping the filter tubes constantly submerged while the thickener is in operation. The filter elements, their supporting frames, the central automatic valve body and the filtrate-collecting receiver are stationary. An automatically operated valve plug in the central valve body and the discharge rakes are the only moving parts.

The tubular elements are of wood, corrugated and slightly tapering. They are covered with a sack of suitable filter fabric. Ordinarily, Genter thickeners have 16 or 32 tube frames, radiating from the central valve body.

In operation, a vacuum pump is connected to the central-filtrate receiver, thus inducing suction on the interior of all filter elements. Solution or filtrate is drawn through the fabric on

the tubular elements and is sent where desired; at the same time, slime builds up on the fabric. After a predetermined time the suction is cut off, and a countercurrent or kickback pressure of compressed air or of filtrate is applied for a few seconds which

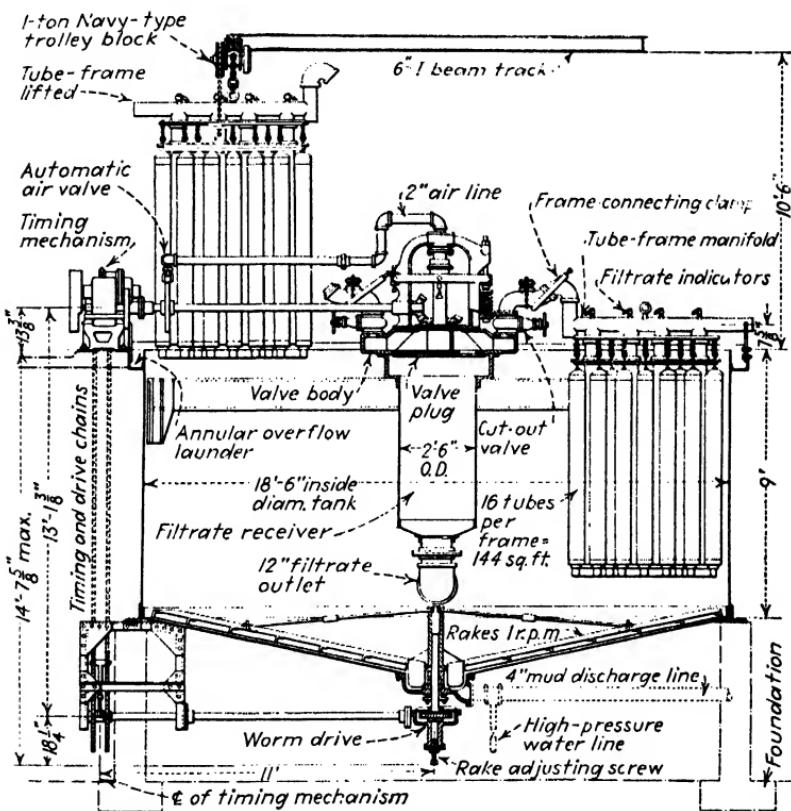


FIG. 44.—Sectional elevation "B-H" type Genter thickener.

removes the cake of slime. The slime cake falls to the bottom of the tank and is raked out.

This thickener has found limited application in metallurgical work because of its relatively high operating cost and the necessity for constant attention while in operation.

FILTRATION

Definition.—Filtration may be defined as the separation of liquids from solids by passing the liquid vehicle through a porous

medium, which offers free passage to the liquid but refuses passage to the solids suspended therein. Either the solid or the liquid constituent or frequently both may be products of value, and the ratio of liquid to suspended solids in the feed pulp may vary between extremely wide limits.

Principles.—The principles of filtration include these considerations: (1) It is evident that a differential in pressure must exist on the two sides of the filter medium. (2) It is rarely true that the filter medium alone is capable of retaining all of the solids during the initial period of filtration; after formation of the initial thin cake operators usually depend on the layer of solids rather than upon the medium for the collection of the solids and clarification of the liquid. (3) Capacity, in terms either of solids filtered or of filtrate clarified, is proportional to the time required for completion of the entire cycle, including cake formation, washing, drying and discharging. These, in turn, are primarily functions of the rate of flow of the liquid through the filter cake. (4) The rate of flow of liquids in filtration is roughly proportional to the pressure differential and roughly inversely proportional to the cake thickness.

In metallurgical work the filter is usually preceded by a thickener to dewater the pulp as far as possible by gravity, thereby producing a uniform feed and one on which the filter can operate at its greatest capacity and give best washing results.

Filtration Media.—These may be divided into two main classes—the fabric and the metallic media. Of the former, the most commonly used are of cotton, different weights of duck, twill cloth and special weaves, sometimes chemically treated for special purposes. Of the metals, there are woven-steel and monel-metal screens and wire. The cotton ducks are satisfactory and are used mainly on pressure-type filters. The twill cloths are resistant to abrasion and are widely used on vacuum filters. The special weaves—chain, square and basket—find their chief application on vacuum filters where high capacities must be obtained with coarse solids and a clear filtrate is not essential. Factors governing the selection of a filtering medium are:

1. It must have mechanical strength and resistance to the solution to be filtered and have a satisfactory economic life. This determines the weight and type of the material.

2. It must produce a filtrate of desired clarity and retain the smallest particle in the feed. This determines the grade and weave of the cloth.

Filter Cloth.—D. F. Irvin of Oliver United Filters contributes the following additional information on filter cloth:

The texture of filter cloth as used in filters in cyanidation for some years has been largely that of so-called twills, and in these cotton fabrics the weight usually runs from 15 to $17\frac{1}{2}$ oz. The texture as such would be difficult to describe except that it produces a diagonal ribbing of characteristic appearance with which all cyanide operators are familiar. This is produced by alternately skipping certain threads in the weaving which on the next threads are not omitted.

At an earlier period the so-called square-weave duck fabrics were used, but they are distinctly unsuited for this work, being better fitted to retain liquids than to permit their passage.

In more recent years there has been a tendency toward lighter material, and with better general understanding of filter fabrics this tendency may proceed still further. Meanwhile, it remains a matter for determination by trial as to the best fabric in a given case.

The price has fluctuated with conditions in the cotton-textile industry since the summer of 1933. Generally, the material is sold in make-up lengths as well as being stitched for a given width of filter in the case of the Oliver machine. Recently, seamless filter cloth up to 14 and 16 ft. in width has been supplied. Much of this has been sold during the past year in the South African cyanide plants, and some in North America. There are certain obvious advantages to a seamless filter cover.

Occasionally, material is sold in rolls for the mining company to make up its own covers, but the customary practice is to buy the cover ready-made. In the case of the American disc filters, which use a specially tailored bag to cover the individual sectors, the covers are made up to fit and are sold by the dozen or other convenient multiple. The bags used on the American filters are occasionally heavier than the covers used on the Oliver filters because at times the wear is greater.

As to the useful period for cloth on cyanide filters, wide variation is reported. The longest life known is that on the Oliver filters at the Hollinger where the cloths now last 220 to 230 days

—a remarkable performance. From this figure of nearly 8 months we go down through various periods, finding that 150 to 160 days is considered excellent life and that a good many plants are obtaining 100 to 120 days. Where working conditions are severe, the life is less than 100 days, but possibly in all such cases means might be found to increase it.

In removing lime encrustations by hydrochloric acid, 5 per cent is considered too concentrated for safety, and 2 to $2\frac{1}{2}$ per cent is preferable, with reasonably frequent treatments, instead of stronger acids, being used at longer intervals.

The frequency of acid treatment is a factor that varies considerably in different places. In Northern Ontario it is used much more often than in other places where lime concentration is less. In this respect it resembles the frequency of acid treatment necessary in the silver cyanide plants in Southern Mexico where extremely high lime concentration prevails.

Stage Filtration.—Two and sometimes three stages of vacuum filters have been used in series with pulping apparatus between each stage for washing pulp. Diluting liquid is added to each filter cake ahead of each repulper. In some cases partial C.C.D. is employed prior to the filtration stage to reduce the value or grade of liquid going to the filters. Sometimes there are short periods of agitation between the filters.

This method of working has been adopted by several of the Canadian cyanide mills, but its use has not been general elsewhere. Operating costs are relatively high compared to C.C.D. or C.C.D. followed by a single dewatering filter. For examples see descriptions of the Lake Shore, Wright-Hargreaves, Teck Hughes and Hollinger cyanide mills in Chap. XII.

Filter Types.—Two types of filters are in use—those that employ pressure and operate intermittently, with definite cycles for charging, washing and discharging; others that employ vacuum and operate continuously or intermittently, with synchronized feeding, cake forming, washing and discharging. Among the filter presses are the Dehne, Shriver, Merrill, Kelly, Sweetland and Burt. Of these in current gold and silver metallurgy the Dehne has some use in Australia and the Merrill in North America and Central America. Among the vacuum filters in use today are the Moore and Butters leaf filter of the

intermittent or cyclic type and continuous revolving types such as the Oliver, the American and the Dorrcos rotary filter.

Pressure Filters. *Dehne Press.*—The Dehne plate-and-frame filter press has satisfactorily filtered many millions of tons of raw and roasted slime in Australia, particularly in the Western state. At Kalgoorlie 100 of them and at one time 80 were treating 100,000 tons a month. Their operation was described in detail by M. W. von Bernewitz in 1906 in *Proc. Australian I.M.E.*, now the *A.I.M. and M.* Most of these presses were of 5-ton capacity per charge. A few were hydraulically closed, but most of them



FIG. 45.—An installation of Oliver filters at the Hollinger Mill, Ontario, Canada.

were hand closed. In their discharge much hand labor was used. They can be emptied slowly by reversing the washing valves and opening the filling valve. A few Dehne presses remain in use today at Kalgoorlie, on roasted ore; all of the others have been replaced by Oliver filters. Pressure filters are expensive in operation, and good results are dependent on careful attention and manipulation on the part of the operator.

Merrill Press.—The Merrill filter press (see Fig. 46) is essentially a plate-and-frame press but is practically automatic in filling and discharge. Unlike the Dehne, the Merrill does not need opening save for renewing cloth or making other minor repairs. Along the median line at the bottom of the press, pass-

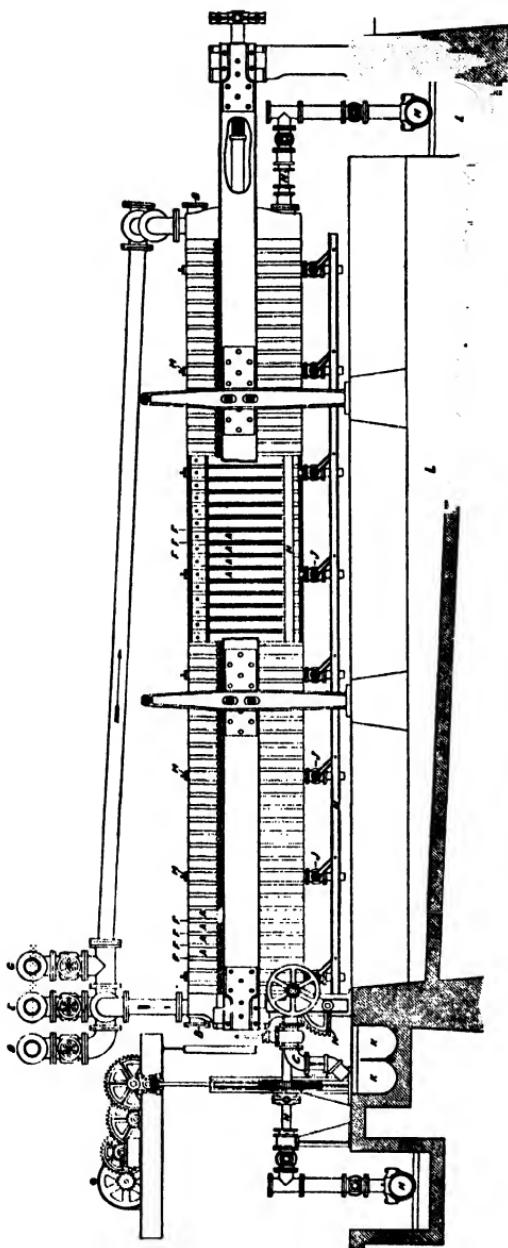


Fig. 46.—Merrill automatic sluicing-pressure slime filter—part-filling or center-washing type.

A. Standard container or frame.
 B. Feed channel through which the slime pulp enters each frame.
 C. Pipe from which water or solution is drawn off during process of filtration.
 D. Slime or pulp line.
 E. Barren-solution wash line.
 F. Filter plate.
 G. Water wash line.
 H. Sluicing pipe, containing water under 60- to 90-lb. pressure admitted at either or both ends.
 I. Sluicing motor and gearing.
 J. Residue discharge cocks.
 K. Effluent solution launders.
 L. Discharged residue launder.
 M. Peep gate.
 N. Lever system for residue discharge cocks.

SLIME-TREATMENT CYCLE AT THE HOMESTAKE

Operation	Period	Remarks
	Hour Minutes	
1. Filling.....	1 25	Discharge from sludge pressure tank to presses contains 35 % solids. Gage pressure, 20 lb. Effluent partly to waste and partly to storage for washing.
2. Aerating.....	.. 30	Gage pressure, 25 lb. Effluent to waste and to storage.
3. Packing.....	.. 15	Effluent to waste and to storage.
4. Strong-solution leaching.....	.. 30	Lixiviant: 0.055 % NaCN; 1.14 gald; alkalinity, 14. Gage pressure, 30 lb. Effluent to low-solution pumps, followed by clarification and precipitation. Barren solution used as preliminary wash.
5. Soaking.....	.. 40	Effluent to low-solution pumps, followed by clarification and precipitation. Barren solution used as preliminary wash.
6. Aerating.....	1 ..	Gage pressure, 25 lb. Effluent to weak-solution sumps followed by clarification and precipitation. Barren solution reused.
7. Strong-solution leaching.....	.. 25	Gage pressure, 30 lb. Effluent to weak-solution sumps, followed by clarification and precipitation. Barren solution reused.
8. Weak-solution leaching.....	1 15	Lixiviant: 0.02 % NaCN; 1 et. gold; alkalinity, 3. Gage pressure, 30 lb. Effluent to weak-solution sumps, followed by clarification and precipitation. Barren solution reused.
9. Low-solution leaching.....	Lixiviant: 0.015 % NaCN; 1 et. gold. Gage pressure, 30 lb. Effluent to weak-solution sumps, followed by clarification and precipitation. Barren solution reused.
10. Washing.....	1 40	Solution displaced during press filling is raised to full cyanide strength in strong-solution sump, then to storage tanks. Wash effluent to low-solution sump; followed by clarification and precipitation. Barren solution used as preliminary wash.
11. Sluicing.....	1 5	Gage pressure, 60 lb. Residue pulp to discharge.
Total cycle.....	8 45	

ing through each plate and frame, is a continuous channel within which is a sluice pipe bearing a series of nozzles, one for each frame. After a cake has been formed and washed, the sluice pipe is oscillated by a small motor through an arc of 180 deg., and water under pressure is discharged from the nozzles. The slime thus washed out of the frame is repulped with the water and discharged from the press through a number of cocks leading from the annular sluicing chamber.

Thirty-one Merrill presses at the Homestake mine, South Dakota, treat—not merely wash and filter—at least 1500 tons a day of slime of which 99 per cent passes 200 mesh. Each of these presses has 90 frames, 6 by 4 ft. by 4 in., the last dimension being the thickness of the slime cake. They hold 26 tons of dry slime. The slime is dewatered at the stamp mills to 35 per cent solids and flows by gravity to the slime-treatment plant three miles distant. The press plates are covered by two cloths—a light muslin twill next to the frame and No. 10 cotton duck which covers the muslin; the muslin prevents the duck from being forced into the plate corrugations. The cotton has a life of 20 months. A typical treatment cycle, as given by A. J. Clark in *E. and M.J.*, Oct. 12, 1931, the Homestake number, is shown in the table on page 125.

The Homestake is the only plant practicing direct treatment in Merrill presses. There are two reasons for this: (1) Few ores are amenable to such treatment; (2) few mines have enough ore in reserve to justify the capital cost of such a plant.

Four other gold and silver plants have installations of Merrill presses—the Dome in Ontario, Canada; the San Luis in Durango and the Santa Gertrudis in Hidalgo, Mexico, and the New York and Honduras Rosario in Honduras. At the Dome mill four Merrill presses operate on the following cycle: 20 min. for filling with treated slime, 50 min. for washing and 50 min. for discharging by sluicing, plus a few minutes for sundry manipulation.

VACUUM FILTERS

Leaf Type. *Moore and Butters.*—The Moore was the first of the leaf filters and was developed in 1901. The individual leaf consists of a frame formed of a perforated pipe through which suction and compressed air are applied. The pipe is covered with a filtering medium. An assembly of such leaves is known as a

"basket." Four tanks having vertical sides part way and then tapering to the bottom are required. The first tank holds the slime to be filtered; the second is filled with weak solution and the third with water for washing the slime cake; the fourth tank is used for receiving the washed slime, loosened from the leaves by compressed air, and repulping it for discharge to the dump. The basket is raised and lowered and moved from tank to tank by means of a chain hoist mounted on an overhead runway extending the full length of the three tanks.

The Butters filter has been used to a greater extent than the Moore, but both have now been largely replaced by C.C.D. and continuous filters such as the Oliver drum type. The Butters filter is similar in construction to the Moore, but the basket always remains in one tank. Into and from this, according to the cycle, are pumped and discharged slime and washing solutions or water. The treated slime is discharged as in the Moore system.

1. Butters filters are used at the Golden Cycle mill, Colorado Springs, for final filtering of roasted slime in the cyanide plant. The use of these filters is described by N. W. Bowen in *E. and M.J.*, August, 1935, and by L. S. Harner in *I.C. 6739, U. S. B. of M.*

There are two filters containing a total of 121 canvas-covered filter frames, $6\frac{1}{2}$ by $9\frac{1}{2}$ ft. According to Harner, about 30 per cent of the mill feed of 1000 tons per day is treated in the slime section of the plant. The slimes usually run about 4 per cent plus 100 mesh and 84 per cent minus 200 mesh. After the thickening and agitation steps, the pulp is fed to the filters at about 50 per cent moisture and is given a barren-solution wash followed by a short water wash. Capacity is approximately about 40 lb. dry slime per sq. ft. canvas per day. The loss of dissolved value in the filter tailing is unusually low, about 0.0012 oz. (0.024 dwt.) per ton, and is to a considerable extent due to the solids' having been roasted.

2. The operation of Butters filters at the Fresnillo Company's mill, Fresnillo, Mexico, is described by W. E. Crawford in *Trans. 112, A.I.M.E.*

Two final agitators serve as storage tanks for four Butters filters, each consisting of 156 leaves, 10 ft. 6 in. by 5 ft. 6 in. The agitators discharge through 20-in. lines, and the flow is

controlled by hydraulic valves from a manifold in the center of the filter house. Each filter box has a separate 18- by 18-in. Gould wet-vacuum pump. A vacuum of 18 to 19 in. of mercury is maintained, atmospheric pressure being 22.5 in.

The total filter cycle takes 100 min.; 17 min. is used for cake forming, 34 min. for barren-solution wash, 24 min. for water wash and 25 min. for filling and discharging. The water wash is made up of treated solution from which the cyanide has been removed in the cyanide plant and sufficient make-up water to maintain the plant balance.

In order to minimize the reducing action of the acid-treated solution from the recovery plant on the soluble silver in the cake, the solution is first made alkaline with lime and then partially aerated by discharging the return solution on to a metal screen over the storage tank.

Thick pulp discharged after the completion of the filter cycle is pumped to a storage sump. As the discharge becomes thin it is then pumped to four Dorr thickeners, 40 by 15 ft. deep. The thickeners' discharge then goes to the storage sump and from there to the tailings dump. The filter leaves are acid treated every 10 days with dilute sulphurous acid made by passing gas from the roasting plant through water. Thirty filter leaves are replaced every month in these four filters and one 156-leaf clarifying filter.

The barren-solution leaching rate is maintained at 2 tons per hr. per box. It has been noted that a higher leaching rate results in a cracked cake while emptying and refilling during the filter cycle. This results in poor washing efficiency because of short circuiting of solution.

Filter capacity is approximately 60 lb. dry slime per sq. ft. canvas per day. About 8 per cent of the solids are plus 100 mesh, and 58 per cent minus 200 mesh.

3. The Butters filter is still largely used in South Africa. A typical example is at the West Springs mines. According to information supplied the author in January, 1935, by J. L. Willey, consulting metallurgist, an average of 3685 tons per day is treated, the ore being ground to 66 per cent minus 200 mesh. After thickening and agitation the pulp is filtered on 490 Butters leaves. A total of 4050 tons of barren solution is used. The filter discharge contains 30 per cent moisture and assays 0.05 dwt.

(0.0025 oz.) dissolved gold and 0.43 dwt. undissolved gold. Capacity of the filters is about 85 lb. dry solids per sq. ft. canvas per day.

Rotary Type. American Filter.—This machine consists of a number (usually six) of parallel, individual, cloth-covered discs mounted on a hollow shaft through which suction and compressed air may be applied. The lower halves of the discs are submerged in the slime tank. The discs are divided into independent

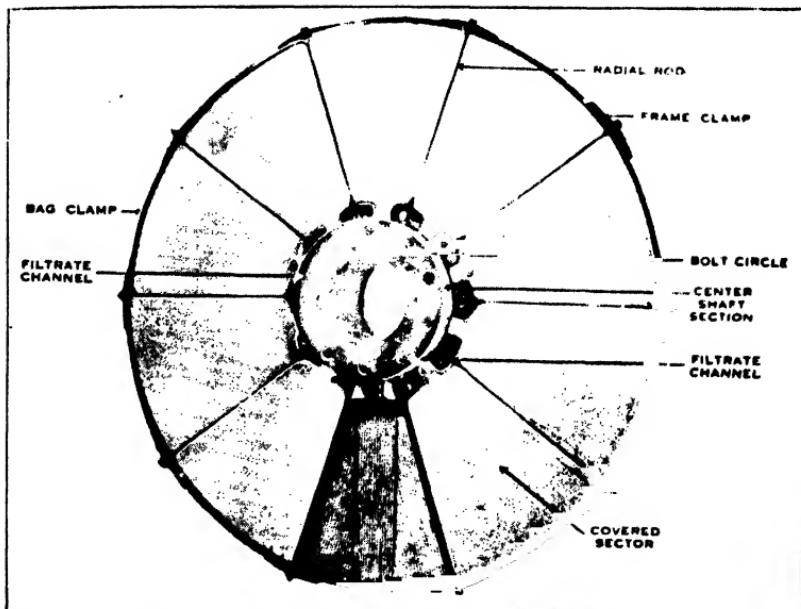


FIG. 47.—Assembled disc and shaft section of American filter.

and removable sectors. Wash water may be applied from spray nozzles. The assembled disc and shaft section is shown in Fig. 47 (see also Fig. 48).

The American filter is a continuous machine of mechanical simplicity. It occupies small floor space and presents a larger filter area or surface in proportion to space filled than any other filter. Inflation of the filter bags during each discharge period keeps the cloth in good condition, and the cake discharged has low moisture content. The changing of filter cloths and the sectors is done easily and quickly. No pulp agitator is needed

in the tank. This filter is made by the Oliver United Filters in four sizes: 4, 6, $8\frac{1}{2}$ and $12\frac{1}{2}$ ft. in diameter with 1 to 12 discs.

In operation the thickness of the cake ranges from $\frac{1}{8}$ to $\frac{3}{4}$ in., depending on the material being filtered. As the disc slowly revolves, each sector in sequence rises from the pulp with vacuum still applied. Fine sprays of solution or water then are applied to the cake, thereby displacing the original liquid left in the pores of the cake. After a sector passes out of the spray zone, vacuum still applied, a large portion of the wash that has replaced the

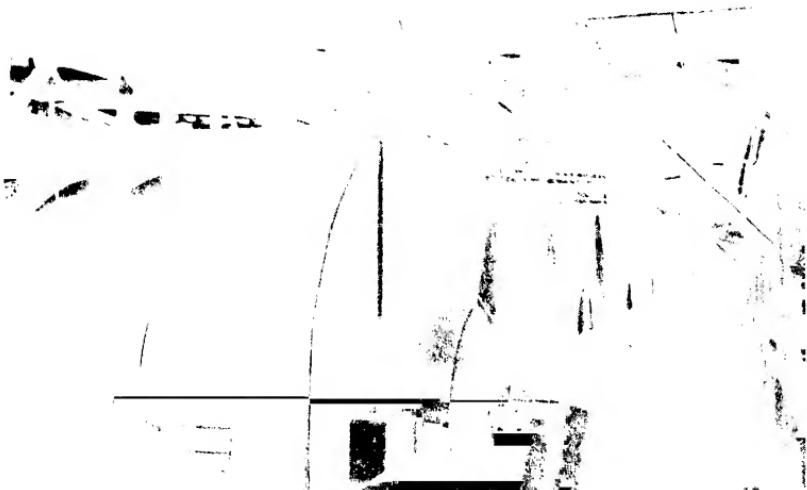


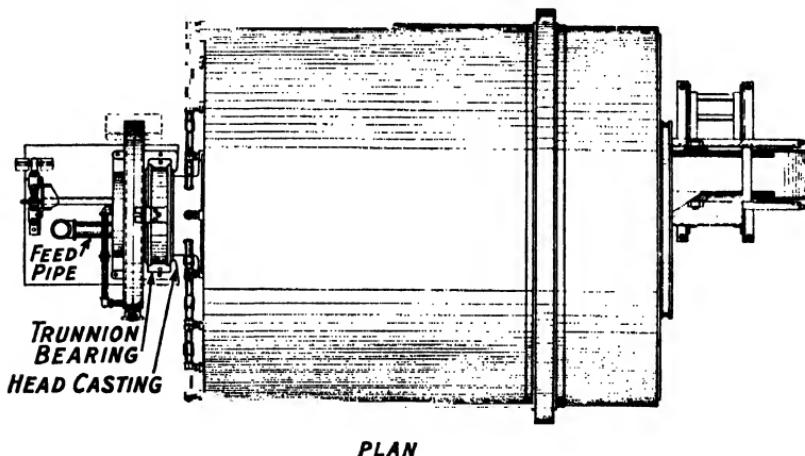
FIG. 48.—American filter installation at Lake Shore, Ontario, Canada.

original liquid is also drawn out. Now as dry as it can be made, the cake is in proper condition for discharging. This is done by cutting off the vacuum and introducing a small quantity of 2 to 3 oz. air, which slightly inflates the bag. The cake will be so loose that it can either drop off or be scraped or rolled off easily.

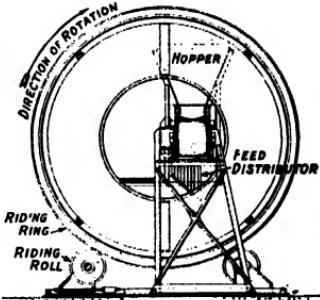
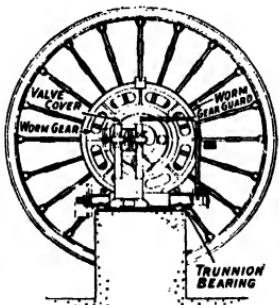
At the Howey mine, Ontario, sector treatment consists of washing, scrubbing with a stiff brush, inspecting for wear and tear and treating in dilute solutions of hydrochloric acid for 5 hr. In this way the cycle of filter-sector treatment is completed in 5 days. Sector bags each handle approximately 500 tons of solids before replacement. The tearing of the bags by scraper

shoes has been the main cause of failure, and it is thought that the rubber-covered shoes which are under test will increase the life of the sector bags substantially. The Howey mill treats 1300 tons of ore daily.

Dorco Filter.—This machine is of the rotary-drum type, but the filtering medium is applied on the interior of the drum



PLAN



Elevation: drive end.

FIG. 49.—The Dorco filter—plan and elevations.

Elevation: cake-discharge end.

which acts as its own pulp container. The inner surface of the drum is divided into a series of fabric-covered panels, each forming a face of a polygon inscribed within the cylinder. The compartments thus formed connect individually with an automatic valve which, in turn, applies vacuum or compressed air in proper sequence. The closed end of the drum is carried by a trunnion and bearing, and the open end is equipped with a

steel tire which runs on rollers. An annular ring forms an internal flange at the open end, serving as a dam for retaining the pulp being filtered. Feed may be introduced either by a pipe passing through the trunnion or by a trough passing through the open end. Cake is discharged by compressed air into an internal hopper, equipped for either chute discharge or discharge on to a conveyor belt. Spray nozzles within the drum may be used for washing the cake. The Dorreo filter in elevation and sectional elevation is shown in Fig. 49. Oliver United Filters makes these machines in six diameters, from 4 to 14 and up to 18 ft. in length.

At the Pioneer mill, British Columbia, the practice in maintaining the Dorreo filter is to treat the filter cloth—a No. 26 twill—with acid every 21 days. The procedure consists of hosing out the filter, dumping in a dilute solution of hydrochloric acid and rotating the filter drum for an hour; the filter is then drained, again hosed out and put into service. There is no scrubbing of the cloth, the pores being kept open easily throughout its life. To keep the cloth in best condition, low-pressure air is blown through it for several seconds after the cake has been discharged. The filter cloth is changed after about 108 days of service; two men make the change in about 8 hr. Each filter has 371 sq. ft. of canvas area, which, on the basis of washing 200 tons of solids per day, shows a capacity of 1078 lb. per sq. ft. canvas per 24 hr.

Oliver Filter. This machine consists of a cylindrical drum divided at its periphery into a number of individual compartments and covered with a cotton material, held in place by a spiral winding of wire. The lower portion of the drum is immersed in a tank filled with thickened pulp to be filtered. The pulp is fed to the tanks continuously and is maintained in suspension by an oscillating paddle. The drum is supported by two trunnion bearings and is revolved slowly by a motor through worm and gear drive. The interior of each compartment of the filter shell communicates through a separate conduit to a rotary port valve, mounted at one end of the drum. Through this valve either suction or positive air pressure is applied to the different compartments in the proper sequence and for the desired period. The slime cake is discharged from the drum by a stationary scraper blade which touches the wire winding around the cloth. Compressed air is admitted at the point of discharge to loosen

the cake. Sprays for washing the cake may be applied through nozzles mounted on the frame. Figure 45 shows some Oliver filters at work.

The Oliver filter is now more generally used than any other type in cyanide plants for washing and final dewatering.

CONTINUOUS COUNTERCURRENT DECANIMATION

Principle.—The principle of continuous C.C.D. is simply that when water or solution is to act upon solids, both are made to pass, in contact, in opposite directions, so that at each end the strongest or most potent portion of either is acting upon the weakest or most exhausted portion of the other.

The recovery of dissolved gold and silver from slime pulp in the cyanide process, as first practiced, employed intermittent decantation.

Attempts to make this process continuous instead of intermittent were made as far back as 1902, but without success. The invention of the Dorr thickener furnished a means of continuous slime settling on a large scale and made the C.C.D. process possible.

Although flow-sheets and operation will vary with conditions, the following will give a general idea of the C.C.D. process.

After the ore has been reduced to a uniform fineness by wet grinding and classification, and the major portion of the solution removed, it is agitated continuously in a series of about three agitators with cyanide solution, where most of the gold and silver is dissolved. The mixture of solids and solution is fed continuously to the first thickener of the C.C.D. series, consisting of from three to five thickeners. It is diluted with the overflow solution from the next thickener in relatively large quantity, a solution that has already been in contact with the pulp in all succeeding thickeners and so is comparatively rich in dissolved gold and silver.

The overflow from the first thickener of the C.C.D. series containing the greatest amount of dissolved metal is sent to precipitation or to the grinding and classifying circuit. The underflow is pumped to the next thickener, diluted with the overflow from the third of the series, settled and thus proceeds through each thickener, until discharged from the last as tailings. By each step of the system a large amount of low-grade solution is

mixed with a very small amount of higher grade solution so that the value of the solution entering each successive thickener is materially reduced. Barren-cyanide solution from the precipitation apparatus is added to the thickener preceding the last and makes up the bulk of the countercurrent flow. Water is added to the last thickener to make up for the liquid losses in the tailing and to cut down cyanide consumption. A thickener called No. 1, precedes the agitators in the mill; therefore the first thickener in the C.C.D. series is usually referred to as No. 2.

Uses.—1. To recover practically all of the dissolved values from the finely ground solids, without filtration. The flow-sheet must be varied somewhat, depending upon the strength of the cyanide solution to be used and the amount of solution required to be precipitated.

2. To reduce the dissolved value in the pulp going to a filter, so that the final dissolved loss will be decreased and the use of a barren-solution wash shortened or omitted. This increases the filtering capacity, diminishes the cost of operating a filter and lowers the loss of dissolved value.

Application.—The application of the C.C.D. process is illustrated by the accompanying typical flow-sheets.

Figure 50 (type *B*) shows the most common type of flow-sheet in use, where the ore settles readily to 50 per cent moisture or less, and where a strong cyanide solution is not required.

The overflow of either thickener *W* or thickener *X* may be used for dilution in the agitators without affecting the distribution of values in the system. By using the overflow of *X* a lower grade of solution is obtained, which may be beneficial, and the dilution of feed to thickener *W* is decreased.

TYPE B

Dissolved Value Loss.

Conditions assumed:

- (a) 100 tons of ore per day crushed in cyanide solutions.
- (b) Discharge from all thickeners with 50 per cent moisture.
- (c) \$10 value dissolved per ton of ore.
- (d) 50 per cent in mill and 50 per cent in agitators.
- (e) 400 tons of solution from thickener *V* precipitated to \$0.02.
- (f) Agitation with a dilution of 2 of solution to 1 of solids.
- (g) Let *V*, *W*, *X*, *Y* and *Z* represent the value in dollars per ton of solution discharged from the respective thickeners.

Equating out of and into Each Thickener:

- (1) $100V + 400V = 500W + (0.50 \times \$10 \times 100)$
- (2) $100W + 600W = 500X + 100W + (0.50 \times \$10 \times 100) + 100V$
- (3) $100X + 500X = 100W + 500Y$
- (4) $100Y + 500Y = 100Z + 100X + (400 \times 0.02)$
- (5) $100Z + 100Z = 100Y + 100 \text{ tons of water value } \0

Simplifying:

- (1) $V = W + 1.00$
- (2) $W = X + 1.20$
- (3) $X = Y + 0.24$
- (4) $Y = 0.2Z + 0.064$
- (5) $2Z = Y$

Solving:

- $V = \$2.51111$
- $W = 1.51111$
- $X = 0.31111$
- $Y = 0.07111$
- $Z = 0.03556$

By calculation, the following results are obtained:

Assay value of the pregnant solution; i.e., value of $V = \$2.51111$.

Assay value of the discharged solution; i.e., value of $Z = \$0.03556$.

Loss of dissolved value per ton of ore, $\$0.03556$.

Dissolved value saved, 99.64 per cent.

Mechanical Loss of Cyanide.

Conditions assumed:

- (a) Neglect the cyanide consumption throughout the system.
- (b) Strength of cyanide per ton of solution, 1.0 lb.
- (c) Let V, W, X, Y and Z represent the strength in pounds of cyanide per ton of solution discharged from the respective thickeners.

Equating out of and into Each Thickener:

- (1) $V = 1.0$
- (2) $100W + 600W = 100W + 100V + 500X$
- (3) $100X + 500X = 100W + 500Y$
- (4) $100Y + 500Y = 100Z + 400V + 100X$
- (5) $100Z + 100Z = 100Y + 100 \text{ tons of water}$

Simplifying:

- (1) $V = 1.0$
- (2) $6W = 5X + 1$
- (3) $6X = W + 5Y$
- (4) $6Y = Z + X + 4$
- (5) $2Z = Y$

Solving:

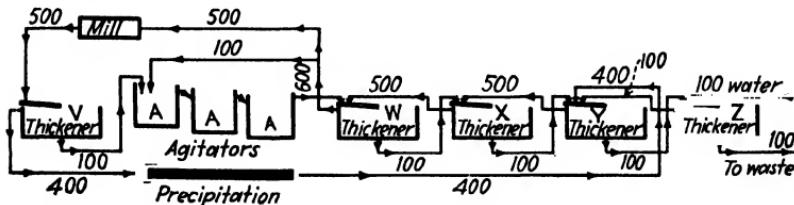
- $V = 1.0$
- $W = 0.9109$
- $X = 0.8932$
- $Y = 0.8898$
- $Z = 0.4449$

$Z = 0.4449$ lb. = mechanical loss of cyanide per ton of ore

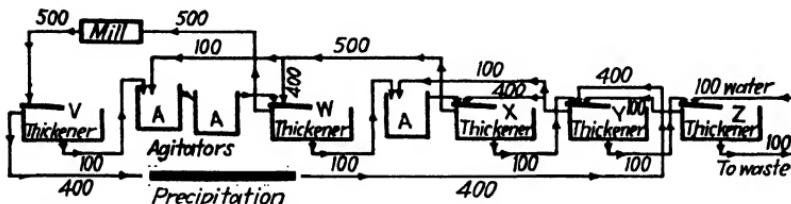
Where the extraction is obtained slowly, and by experiment it is found that an additional change of solution during agitation is beneficial, the flow-sheet shown in Fig. 50 (type BB) may be used. This arrangement allows two changes of solution with a thorough washing of the pulp during agitation. The pulp during

the final agitation is in contact with fresh solution carrying low values, conditions most favorable for dissolving the refractory material in the residues.

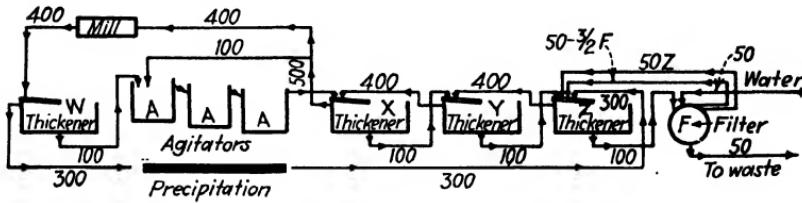
Type D flow-sheet (Fig. 50) is more generally used in cyanide plants today with or without a change of solution during agitation as in type BB. This type of flow-sheet is indicated where solutions stronger than $1\frac{1}{2}$ lb. NaCN per ton are used, where the



Type B.



Type BB.



Type D.

FIG. 50.—Countercurrent decantation; typical flow-sheets.

pulp will not settle to at least 50 per cent solids or where a dewatered filter cake is desirable for tailing-disposal reasons.

TYPE D

Dissolved Value Loss.

Conditions assumed:

- (a) 100 tons ore per day crushed in cyanide solution.

- (b) Discharge from all thickeners with 50 per cent moisture.
- (c) \$10 value dissolved per ton ore.
- (d) 50 per cent in mill, and 50 per cent in agitators.
- (e) 300 tons solution from thickener W precipitated to \$0.02.
- (f) Agitation with a dilution of 2 of solution to 1 of solids.
- (g) Displacement efficiency of filter, 60 per cent; that is, 60 per cent of the value of the solution in the solid cake, which is assumed to contain $33\frac{1}{3}$ per cent moisture, or 50 tons of solution to 100 tons of solids, is recovered. The 50Z returned from the filter to the last thickener represents 50 tons of solution removed in loading the filter, which will, of course, still have the value of Z.
- (h) Let W , X , Y and Z and F represent the value in dollars per ton of solution discharged from the thickeners and filter, respectively.

By calculation, the following results are obtained:

- (1) $W = \$3.3439$
- (2) $X = 2.0939$
- (3) $Y = 0.5314$
- (4) $Z = 0.1408$
- (5) $F = 0.0563$

To check these figures:

Amount precipitated from 300 tons at $(\$3.3439 - \$0.02)$ =	\$ 997.17
Amount lost in tailings, 50 tons at \$0.0563 =	2.815

	\$ 999.985
Amount due to neglected decimals	0.015

Amount dissolved 100 tons at \$10 = \$1000	\$1000.000

From the foregoing the following results are deduced:

Assay value of the pregnant solution, i.e., value of $W = \$3.3439$
 Assay value of the discharged solution, i.e., value of $F = \$0.0563$
 Loss of dissolved value per ton of ore, $5\%_{100} F = \$0.02815$
 Dissolved value saved, 99.72 per cent

Mechanical Loss of Cyanide.

Conditions assumed:

- (a) Neglect the cyanide consumption throughout the system.
- (b) Strength of cyanide per ton of solution in thickener W , 4.0 lb.
- (c) Let W , X , Y , Z and F represent the strength in pounds of cyanide per ton of solution discharged from the thickeners and filter, respectively.

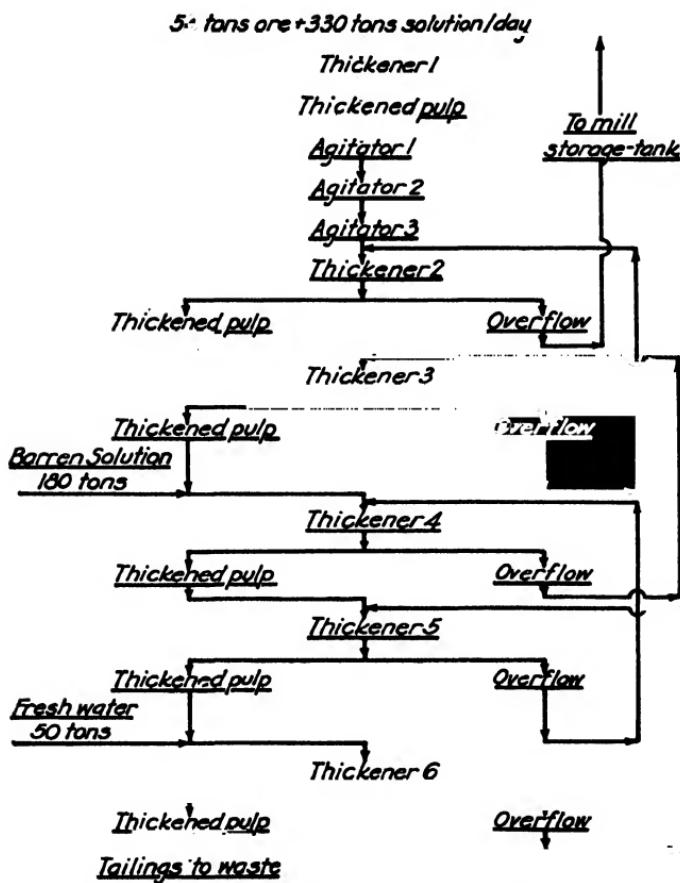


FIG. 51.—Flow of ore pulp and solution in decantation plant of Big Jim mine, Arizona.

By calculation the following results are obtained:

- (1) $W = 4.0$
- (2) $X = 3.8124$
- (3) $Y = 3.7655$
- (4) $Z = 3.7537$
- (5) $F = 1.5015$

$$5 \frac{1}{100} F = 0.7507 = \text{Mechanical loss of cyanide per ton of ore.}$$

Dissolution During Washing.—Some additional dissolving of gold and silver takes place during the washing of the ore pulp after agitation with cyanide solution, either in C.C.D. thickener series or in filters. This is generally credited to the change in

solution which takes place during the washing operation, whereby solution of lower gold and silver content and more freshly precipitated is brought into contact with the ore. Further appreciable time of contact is provided in the C.C.D. thickeners.

Representative data are not available covering additional dissolution in filters, but this does take place to some extent even on continuous rotary filters, where the time of contact is relatively short.

Typical examples of C.C.D. in plant operation may be found in Chap. XII, under descriptions of the Big Jim plant at Ontman, Ariz., and the Pioneer in British Columbia, The United Eastern, Arizona, and the Golden Queen, Calif.

DISSOLUTION IN DECANTING THICKENERS
C.C.D. Cyanide Plants

Operation	Hollinger ^a	United Eastern ^b	Elko Prince ^{c,d}	Mogul ^e
Tons treated per day.....	250	253	53	100
Hours of agitation.....	12	62	72	24
Change of solution during agitation series.....	No	No	Yes	No
Head value to plant:				
Au.....	\$9.00	\$20.66	\$15.27	\$3.10
Ag.....	10.7	1.15
Total.....	\$9.00	\$20.66	\$25.97	\$4.25
Dissolved in:				
Thickener 1, Au.....	\$0.08	\$0.20	\$0.11	\$0.15
Ag.....	0.08	0.03
Total.....	0.08	0.20	0.19 ^f	0.18
Thickener 2, Au.....	0.04	0.07	0.09	
Ag.....	0.02	
Total.....	0.04	0.07	0.11	
Thickener 3, Au.....	0.03	0.04	0.05 ^f	
Ag.....	0.05	0.03
Total.....	0.03	0.04	0.10	0.03
Thickener 4, Au.....	0.03	0.02	No	0.01
Ag.....	0.04
Total.....	0.03	0.02	0.05
Total dissolved in C.C.D. series....	\$0.18	\$0.33	\$0.40	\$0.26

In the foregoing, gold values are based on \$20.67 per oz. and silver on \$1 per oz.

^a From unpublished data, 1916, results in one unit of the C.C.D. series.

^b From A.I.M.E., August, 1919.

^c From A.I.M.E., August, 1918.

^d Third thickener (Elko Prince) followed by dewatering drum filter.

^e Thickener between second and third agitators.

^f From unpublished data from management, 1913.

CHAPTER IX

CONCENTRATION OF ORES AND TREATMENT OF CONCENTRATES

This chapter is divided into three main parts: gravity concentration, flotation and treatment of concentrates.

GRAVITY CONCENTRATION

Gravity concentrators are of the table and endless-belt or vanner types. The former generally consists of a head-motion mechanism which imparts a rapid reciprocating motion to a slightly inclined table with riffles, the slope of which can be adjusted. Richards, in his *Ore Dressing*, gives the following description of a vanner:

A vanner does its work on the upper surface of an endless belt, which is slightly inclined from horizontal and receives a rapid shake in the plane of the belt while at the same time it has a continuous, slow motion up the slope. The travel of the belt carries the heavy mineral to the upper end, and the surface water washes down the light mineral to the lower or tail end. There are two chief classes of vanners, and a third has been recently introduced.

1. Side shake, vibrates at right angles to the direction the belt travels.
2. End shake, vibrates parallel to the direction in which the belt travels.
3. Gyrating.

The table is suitable for a large quantity of relatively coarse feed, whereas the vanner is better adapted to concentrating fine material.

If tables are operated early in the flow sheet, the question arises as to the value of classification of the pulp before tabling. A. W. Fahrenwald discusses this point in *T.P. 403, U. S. B. of M.*, 1927. He concludes that preliminary classification, particularly if classifiers of the hydraulic types are used, gives increased recovery, a higher grade concentrate, greater table capacity and

less middling for regrind; in other words, classification enables a table to do its best work.

Fahrenwald hydraulic classifiers, Richards pulsator classifier and others employing classifying zones of uniform cross section give efficient results.

Concentrating on the Rand.—In some plants the sulphides are separated from the pulp in the Johnson concentrator, which is a rifled, rotating, rubber-lined drum, 30 in. in diameter and 12 ft. long. It is set at a 6 per cent slope and makes 7 r.p.m. A water spray removes adhering concentrate. This product is run over corduroy tables for the recovery of coarse gold, and the tailing rejoins the mill circuit.

Concentration at Argonaut, Calif.—Vanner practice at the Argonaut mine, Mother Lode, California, is described by S. E. Woodworth in *I.C. 6476, U. S. B. of M.*, 1931.

SCREEN ANALYSES OF MILL PRODUCTS FOR A MONTH AT THE ARGONAUT

Sieve mesh		Weight per cent					
		Copper-plate discharge, feed to Spitzlutta classifier	Classifier spigot 1, coarse feed to vanners	Classifier spigot 2, medium feed to vanners	Classifier overflow, feed to Callow cone	Callow cone underflow, fine feed to vanners	Mill tailing
Plus	28.....	0.7
	35.....	9.97	2.30	8.2
	48.....	17.09	9.29	13.8
	65.....	22.92	12.28	4.22	18.9
	100.....	19.31	16.42	14.25	15.7
	200.....	15.11	17.57	41.05	6.06	14.20	23.0
Minus	200.....	15.60	42.13	40.50	93.94	85.80	19.7

The concentrating equipment for each 10 stamps consists of one Spitzlutta-type classifier, six 6-ft. Johnson or Frue vanners and one 8-ft. Callow cone equipped with a gooseneck discharge. The Spitzlutta sizer is a two-spigot machine of local design and manufacture.

The pulps from two five-stamp batteries unite in one launder and drop into one end of the Spitzlutta classifier. The slime overflows from the opposite end and is conveyed to the 8-ft. Callow cone. The first spigot discharge of the Spitzlutta classifier is split between two of the first-row vanners, which handle the coarsest material. The second spigot discharge is split between two of the second-row vanners, which treat the intermediate size of material. The gooseneck discharge of

the Callow cone is split between the two third-row vanners, which handle the finest material. The overflow of the Callow cone passes to the general mill tailing.

Each vanner operates at 180 oscillations per min., has a belt speed of about 4 ft. per min. and treats 8 tons of solids in 24 hr. The proportion of vanner concentrates recovered from the ore treated is variable, depending upon the type of ore. It varies from 1.96 to 3.5 per cent, averaging $2\frac{1}{2}$ per cent. Vanner concentrates have assayed from \$52 to \$135 per ton, latterly around \$60.

The overflow of the Callow cone and the tailings from the three rows of vanners have made up the general mill tailings. If, however,

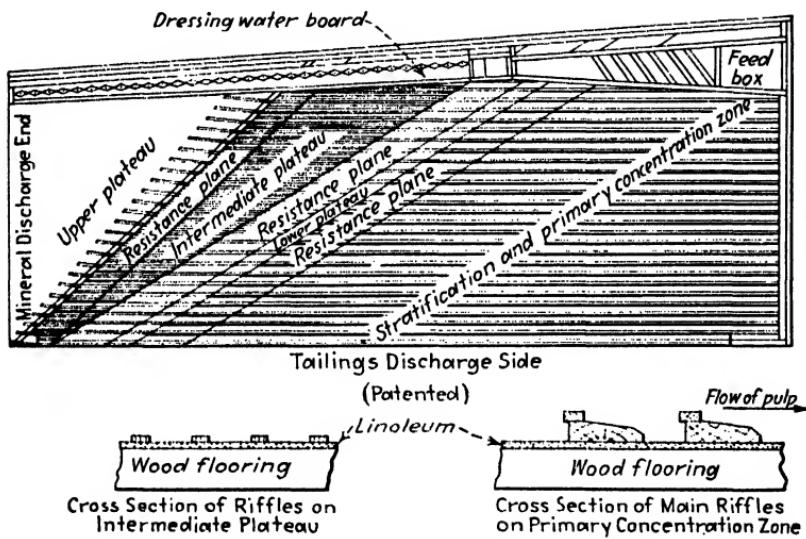


FIG. 52.—Details of Deister concentrator.

value of the general tailings warrants further grinding, the tailings from the first two rows of vanners that treat the coarser products are classified in a Dorr duplex classifier, operated in open circuit with two tube mills. After passing over shaking amalgamating plates the tube-mill product is concentrated on Wilfley tables. The overflow pulp from the Dorr classifier is thickened in a Callow cone, the underflow of which is concentrated on Deister slime tables. Tailings from the Wilfley and Deister tables, the overflows of the thickening cones and tailings from the third-row vanners comprise the general mill tailings which pass to the Amador Metals Reduction Company for cyanide treatment.

The belt machine, or vanner, has always been the standard on the Mother Lode. Various types of shaking sand and slime tables have

been tried, but as a rule they have been rejected for the vanner because it is difficult for mill foremen who have grown up with vanner milling practice and have no first-hand knowledge of other means of concentration to realize that methods and costs have improved with time. Concentration must be employed with these ores, for the gold contained in the sulphide minerals accounts for about 20 per cent of the total value. Flotation heretofore has been of doubtful advantage, owing to the large bulk and low grade of concentrates obtained. This is because of the carbonaceous and colloidal contents of the slime.

Tabling at Presidio, Tex.—The ore from the Presidio mine consists of argentiferous lead minerals, chiefly cerrusite and galena and occasionally some anglesite and the silver minerals, argenteite and cerargyrite, in a gangue of quartz and calcite. Although tabling is used primarily to recover the lead, its removal before the pulp is cyanided results in a lower consumption of cyanide and an increase in the silver recovery.

The ore from the crushing plant is conveyed to a Hum-mer vibrating screen, the undersize of which is treated on the first set of two No. 6 Wilfley tables (see Fig. 53). The feed to these tables is minus-10 mesh material averaging 378 dwt. silver per ton and 3.5 per cent lead; it is maintained at about 14 per cent solids. The tables run at a speed of 250 strokes per min., with a $\frac{3}{4}$ -in. stroke. The maximum capacity of these tables on this type of feed is about 15 tons each per day. The purpose of this tabling operation is to recover as much as possible of the lead before it enters the tube mill and thus avoid excessive sliming. As 50 per cent of the total lead is contained in the feed to these two tables, they play an important part in lead recovery. Each table makes a concentrate, a middling and a tailing product.

The oversize from the Hum-mer screen is fed to a Dorr classifier operating in closed circuit with a tube mill, a stationary screen and a second set of six No. 6 Wilfley tables which also operate at 250 strokes per min. but with a $\frac{5}{8}$ -in. stroke. The tube-mill discharge contains 76 per cent solids but is diluted to 46 per cent for tabling by using a part of the classifier overflow as diluent. Better results would be obtained if the table feed could be diluted more, but existing conditions do not permit more solution to be introduced into the tube-mill circuit. Barren solution is used for headwater on the tables. The stationary screen which receives the tube-mill discharge has $\frac{1}{8}$ - by $\frac{1}{2}$ -in. apertures and

is set at an angle of 50 deg. from the horizontal. The undersize from the screen comprises the feed to the second set of tables; it assays 116 dwt. silver per ton and 0.97 per cent lead, and 39 per cent of it passes 200 mesh. Much of the lead is so badly slimed that recovery is not high. The tables in this set also make a concentrate, a middling and a tailing product.

The middlings from both sets of tables go to one No. 6 Wilfley table which produces a clean concentrate and a tailing. The



FIG. 53.—Strata on a Wilfley table in action.

tailings from all tables are pumped to the Dorr classifier which feeds the tube mill; they assay 70 dwt. silver per ton and 0.7 per cent lead. These tailings after regrinding are cyanided for the recovery of the remaining silver. (See Chap. XII for description.) The combined concentrates assay 8520 dwt. silver and 52.5 per cent lead; they are sun-dried to 10 per cent moisture, sacked, sampled and sent to the El Paso smelter.

The foregoing notes are from the paper by V. D. Howbert and F. E. Gray in *T.P. 368, A.I.M.E.*, 1930, and brought up to date by M. C. Haas, mill superintendent.

FLOTATION OF PRECIOUS-METAL ORES

Precious-metal ores that are amenable to flotation may be classified briefly as follows:

1. Gold ores in which a part of the gold is in the form of fine free gold, the remainder associated with sulphides. Pyrite, arsenopyrite and the sulphides of copper are the common auriferous sulphides. In many ores these sulphides have undergone oxidation. This type of ore commonly contains only minor amounts of silver.



FIG. 54.—Denver "Swb-A" Fahrenwald flotation machine, manufactured by Denver Equipment Company.

2. Silver ores containing argentiferous galena or silver sulphides, sulphantimonides and sulpharsenides such as argentite, pyrargyrite, proustite, jamesonite and tetrahedrite. If ores of this type have undergone oxidation, cerrusite, anglesite and cerargyrite may be present in appreciable quantity. If pyrite is present, it is generally auriferous.

Free gold does not float so readily or so rapidly as most of the mineral sulphides; therefore, if the proportion of gold in the flotation feed is high, a longer period of flotation may be required. With the ordinary ore of milling grade the time factor is not so important. As brought out by A. M. Gaudin in his book

Flotation, it is difficult to reduce coarse gold flakes to a floatable size because grinding does not comminute the gold particles but simply flattens them. Furthermore, this could be an uneconomical procedure in view of the readiness with which coarse free gold is recovered by amalgamation or gravity concentration.

For the treatment of silver ores Gaudin also observes that flotation has been of greater importance partly because silver is generally well recovered by flotation and also because cyanidation and amalgamation are not so effective in extracting silver as in extracting gold.

Flotation Machines.—Many types of flotation have been developed—straight air, mechanical and combinations of the two.

REAGENTS USED IN FLOTATION OF GOLD ORES

Conditioning agents are commonly used, especially when the ores are partly oxidized. Soda ash is the most widely used regulator of alkalinity. Lime should not be used because it is a depressor of free gold and inhibits pyrite flotation. Sodium sulphide is often helpful in the flotation of partly oxidized sulphides but must be used with caution because of its depressing action on free gold. Copper sulphate is frequently helpful in accelerating the flotation of pyrite and arsenopyrite. In rare instances sulphuric acid may be necessary, but the use of it is limited to ores containing no lime. "Ammo-phos," a crude monoammonium phosphate, is sometimes used in the flotation of oxidized gold ores. It has the effect of flocculating iron oxide slime, thus improving the grade of concentrate. Sodium silicate, a dispersing agent, is also useful for overcoming gangue-slime interference.

Promoters or Collectors.—The commonly used promoters or collectors are "Aerofloat" reagents and the xanthates. The most effective promoter of free gold is Aerofloat flotation reagent 208. When auriferous pyrite is present this reagent and reagent 301 constitute the most effective promoter combination. The latter is a higher xanthate which is a strong and nonselective promoter of all sulphides. Amyl and butyl xanthates are also widely used. Ethyl xanthate is not so commonly used as the higher xanthates for this type of flotation.

The liquid flotation reagents such as Aerofloat 15, 25 and 31 are commonly used in conjunction with the xanthates. These

reagents possess both promoter and frother properties. When malachite and azurite are present, reagent 425 is often a useful promoter. This reagent was developed especially for the flotation of oxidized copper ores.

The amount of these promoters varies considerably. If the ore is partly oxidized, it may be necessary to use as much as 0.30 to 0.40 lb. of promoter per ton of ore. In the case of clean ores, as little as 0.05 lb. may be enough. The promoter requirement of an average ore will usually approximate 0.20 lb.

Frothers and Froth Modifiers.—The commonly used "frothers" are steam-distilled pine oil, cresylic acid and higher alcohols. The third mentioned, known as duPont frothers, have recently come into use. They produce a somewhat more tender and evanescent froth than pine oil or cresylic acid; consequently they have less tendency to float gangue, particularly in circuits alkaline with lime. The duPont frothers are highly active frothing agents; therefore it is rarely necessary to use more than a few hundredths of a pound per ton of ore.

When coarse sulphides and moderately coarse gold (65 mesh) must be floated, "froth modifiers" such as Barrett Nos. 4 and 634, of hardwood creosote, are usually necessary. The function of these so-called froth modifiers is to give more stable froth having greater carrying power.

REAGENTS FOR SILVER ORES

The conditioning agents used for silver ores are the same as those for gold ores. Soda ash is a commonly used pH regulator. It aids the flotation of galena and silver sulphides. When the silver and lead minerals are in the oxidized state, sodium sulphide is helpful, but it should not be added until after the sulphide minerals have been floated, because sodium sulphide inhibits flotation of the silver sulphide minerals.

Aerofloat 25 and 31 are effective promoters for silver sulphides, sulphantimonides and sulpharsenides, as well as for native silver. When galena is present, No. 31 is preferable to No. 25 because it is a more powerful galena promoter. Higher xanthates, such as American Cyanamid reagent 301 and amyl and butyl xanthates, are beneficial when pyrite must be recovered. When the ore contains oxidized lead minerals, such as anglesite and cerussite, sodium sulphide and one of the higher xanthates may be used.

In some instances reagent 404 effects high recovery of these minerals without the use of a sulphidizing agent.

Silver ores require the same frothers as gold ores—*viz.*, pine oil, cresylic acid or duPont frothers.

(The foregoing notes were submitted by S. J. Swainson of the American Cyanamid Company. "Aero," "Ammo-phos" and "Aerofloat" are registered trade-marks applied to products manufactured by this company.)

The Great Western Electro-Chemical Company, California, makes amyl xanthate, butyl xanthate, potassium xanthate and sodium xanthate. In the United States these reagents are used on the gold ores of California and Colorado and in Canada on the gold ores and sulphides of Ontario and Quebec.

Flotation reagents of the Naval Stores Division of the Hercules Powder Company are as follows: Yarmor F pine oil, a frother for floating simple and complex ores; Risor pine oil, for recovering sulphides by bulk flotation; Tarol 1, a toughener of froth, generally used in small amount with Yarmor F, but with some semi-oxidized ores where high recovery is essential yet the grade of concentrate not so important, Tarol does good work; Tarol 2, a frother for floating certain oxide minerals, but it can be used in selective flotation of sulphide minerals and in bulk flotation where tough froth is desirable; Solvenol, for the floating of graphite in conjunction with Yarmor F.

PRECIPITATING EFFECT OF FLOTATION REAGENTS

The statement has come to the attention of the American Cyanamid Company that organic flotation reagents, such as xanthates, even in the small amounts used in flotation, cause reprecipitation of gold from pregnant cyanide solutions. The ore-dressing laboratory of this company is studying the question, and preliminary results indicate that this statement is unfounded. The addition of xanthate, in the amount usually found in flotation circuits, does not precipitate gold from a pregnant cyanide solution containing the normal amount of cyanide and lime.

PRIMARY SLIME

Valueless slime, in addition to its detrimental effect in coating gold-bearing sulphide, thereby limiting or preventing its flotation, also becomes mixed with the flotation concentrate and lowers

its value. Sometimes the problem in flotation is that although the gold is floatable, the concentrate product is of too low grade. Talc, slate, clay, oxides of iron and manganese or carbonaceous matter in ores early form slime in a mill, without fine crushing. Such "primary slime," according to E. S. Leaver and J. A. Woolf of the U. S. Bureau of Mines, interferes with the proper selectivity of the associated minerals and causes "slime interference." The tendency of primary slime is to float readily or to remain in suspension and be carried over into the concentrate. Preliminary removal and washing of this primary slime before fine crushing is one method of dealing with it. At the Idaho-Maryland mill, Grass Valley, Calif., starch is regularly used as a depressant during flotation. Flotation tests using starch were made on a quartz ore containing carbonaceous schist from the Argonaut mine, Jackson, Calif.; a talcose ore from the Idaho-Maryland mine mentioned; a talcose-clayey ore from Gold Range, Nev.; a siliceous, iron and manganese oxide ore from the Baboquivari district, Nevada; carbonaceous and aluminous slime from the Mother Lode and some synthetic ores. The conclusions from the foregoing tests were in part:

1. Finely divided metallic gold in milling ores floats readily, and a high-grade concentrate can be made by flotation if no interfering slime or gangue is present. Any good collector may be used for the flotation of gold, but organic collectors of the xanthate type produce a cleaner, higher grade concentrate than coal tar-creosote oils.
2. Some "protective colloid" should be added to "wet out" talcose or carbonaceous slime and destroy its tendency to float.
3. Clayey slime does not have strong, flotative properties, but it tends to remain in suspension and coat mineral particles, making it difficult to obtain good selectivity during flotation. Proper deflocculation of an ore pulp and some agent to destroy the flotative property of clay improves flotation of this type of ore.
4. It is essential to keep pulp containing iron (not hematite) and manganese oxides in a dispersed condition, and improved results are obtained by means of a depressing agent such as starch.
5. Starch was the most effective depressing agent tried. It should be added as a solution to the ore pulp. Starch displays a selective action in its depressing effect. It acts first on the slime; then, if a sufficient excess of starch is present, it will cause some depression of sulphides and metallic gold, either by wetting out or by producing an extremely brittle froth. Therefore, care must be taken in regulating

the amount of starch added to obtain the maximum depression of the slime commensurate with high recovery of the gold. In this, as in all other phases of flotation, each ore presents an individual problem and must be so studied.

GOLD LOSSES IN FLOATING COPPER, LEAD OR ZINC SULPHIDES

Sodium cyanide at the rate of 0.03 to 0.50 lb. per ton of ore is generally used in the differential flotation of complex sulphide ores. If the pulp density in the grinding circuit be 75 per cent solids, this amount of cyanide is equivalent to 0.09 to 1.5 lb. per ton of solution which corresponds to the solution strength commonly used for the cyanidation of gold ores.

If the complex ore contain gold, the effects of the use of cyanide as a depressant should be investigated.

A. K. Anderson, of the Canadian Department of Mines, as reported in *C.M.J.*, June, 1934, reviews the results of tests made on an auriferous complex ore. The ore assayed: gold, 0.59 oz.; silver, 3.4 oz.; lead, 8.4 per cent; zinc, 9.1 per cent. Cyanide at the rate of 0.30 lb. per ton of ore was added to the grinding circuit. The dilution in the grinding circuit was 66 per cent solids and in flotation about 30 per cent solids. A lead concentrate and a zinc concentrate were made. Filtrates from the concentrates and the tailing were combined and assayed for gold. It was shown that nearly 11 per cent of the gold in the feed remained in solution, equivalent to 0.06 oz. per ton of ore.

Methods for recovering the soluble gold are outlined.

CHECKING FLOTATION BY TABLE CONCENTRATION OF TAILINGS

In a small flotation plant it probably would be an unnecessary expense to operate a pilot table concentrator to check the results of flotation, but in a large plant it is advisable. Even there, passing the whole flow over one or more tables is not necessary; a continuous sample of the final residue, say 2 per cent of the pulp, is ample to verify the efficacy of flotation.

CYANIDATION VS. FLOTATION

The factors to be considered in deciding between cyanidation and flotation for treatment of precious-metal ores are summarized in *Ore-dressing Notes* of the American Cyanamid Company, for March, 1935.

The first and most important consideration in the choice of process is the net profit that it will yield. As a rule, the initial

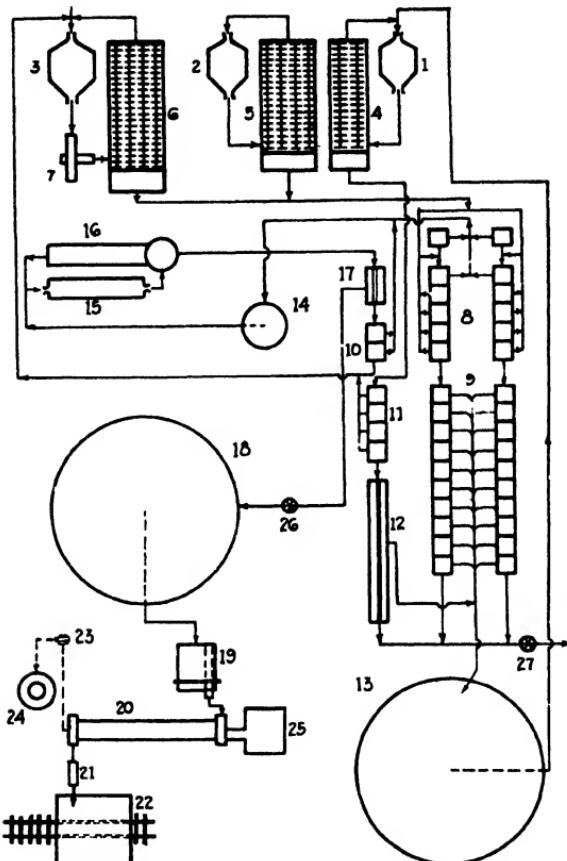


FIG. 55.—Flow-sheet of Beattie plant; milling and flotation.

- 1, 2, 3. Hardinge ball mills (8 by 60; 8 by 72; 10 by 66).
- 4, 5, 6. Dorr classifiers (8 by 25; 12 by 28; 12 by 32).
- 7. Scoop elevator.
- 8, 9, 10, 11. Denver flotation equipment (6-cell primary machines in parallel; 10-cell rougher machines in parallel; 2-cell cleaner; 4-cell middling cleaner).
- 12. Southwestern air-cell cleaner.
- 13. Dorr 40-ft. middling thickener.
- 14. Thickener.
- 15. Tube mill.
- 16. Dorr simplex bowl classifier.
- 17. Southwestern find cleaner.
- 18. Dorr concentrate thickener.
- 19. Dorrco 8 by 10-ft. filter.
- 20. Ruggles-Coles single-shell dryer.
- 21. Bucket elevator.
- 22. Storage bin.
- 23. Fan.
- 24. Cyclone arrester.
- 25. Furnace.
- 26. Concentrate sampler.
- 27. Residue sampler.

cost of building a flotation plant is considerably less than the initial cost of a cyanide plant of the same capacity. Likewise,

the operating cost of flotation is usually lower than that of cyanidation. It should be remembered, however, that in the case of a flotation plant the precious metals are recovered in the form of a concentrate which either must be shipped to a smelter or cyanided on the property.

Although the art of floating precious-metal ores has improved considerably the last few years, it is only fair to point out that the process is by no means universally applicable to the treatment of gold and gold-silver ores. But if precious-metal ores contain cyanicides, which preclude the use of cyanidation, flotation may be the only solution of the problem. With such ores it is often possible to use cyanidation after the cyanicides have been removed by flotation. In this case flotation would constitute an accessory process.

In summation it may be said that cyanidation is still the most universally applicable process in the treatment of precious-metal ores. The present indications are that flotation will serve its most useful function in the role of an accessory process in conjunction with cyanidation.

Fine Grinding and All-cyanidation vs. Fine Grinding and All-flotation.—A noticeable trend in ore treatment is fine grinding and flotation, followed by treatment of the concentrate only. This means that at least nine-tenths of the ore is discarded, and one-tenth treated. Proponents of this process are the Great Boulder, Lake View and Star, and Wiluna in Australia, McIntyre-Porcupine in Ontario, Beattie in Quebec, and Cam and Motor in Rhodesia. If precious metals be so encased in the gangue that only excessive and expensive grinding will free them, cyanidation must be considered. In some plants mentioned, the ore formerly was all-cyanided.

The suitability of the method involving fine grinding and flotation with treatment of the concentrate and rejection of the remainder should receive careful study in the laboratory and in a pilot plant. McIntyre-Porcupine ran a 150-ton plant for a year before deciding to build its 2400-ton mill. Comparative figures given by J. J. Denny in *E. and M.J.* November, 1933, on the results obtained by the all-sliming, C.C.D. process formerly used and the present combination of flotation and concentrate treatment showed a saving of 12.1 cts. per ton in treatment cost and a decrease of 15 cts. per ton in the residue, a total of 27.1 cts. per ton in favor of the new treatment.

T. B. Stevens, in *M.M.*, October, 1933, gave current treatment at the Lake View and Star in the form of extraction or statistical flow-sheet, which is reproduced in Fig. 58. He gives the values in shillings per ton, but to save using sterling and converting them to dollars they are shown as so many units, which are equivalent. The total cost of all-sliming and flotation treatment

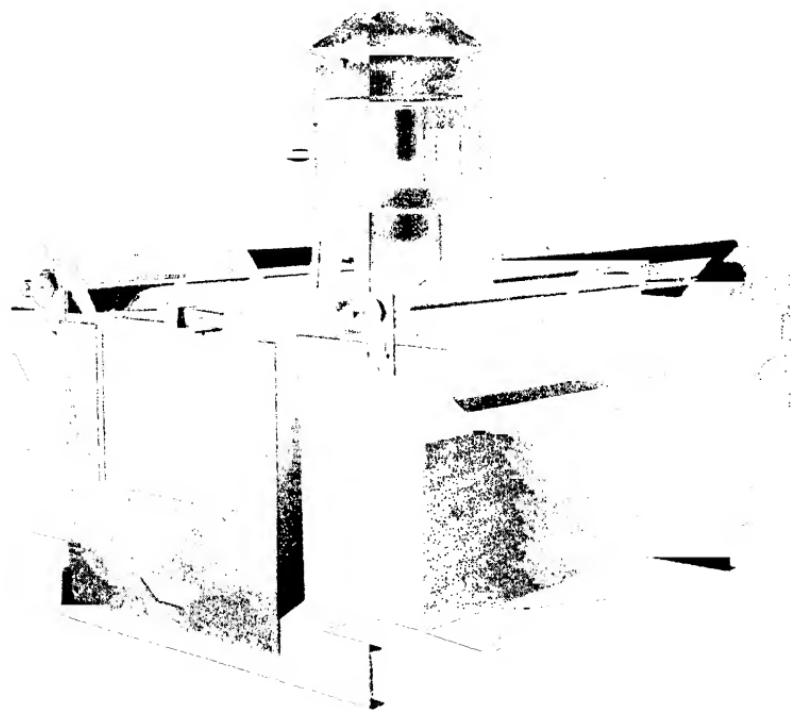


FIG. 56.—Standard square Fagergren flotation machine, manufactured by American Cyanamid Company.

of 45,000 tons a month in 1933 was 6.3 shillings (\$1.52) per ton, compared with 16 shillings (\$3.84) when all-dry crushing, roasting and cyaniding 15,000 tons a month. This use of flotation was an aid in solving the treatment problems at the Kochkar mines, 50 miles southeast of Cheliabinsk, southern Russia, according to M. C. Haas in *E. and M.J.*, October, 1932. One mine yields ore consisting of quartz and some schistose material carrying 2 per cent pyrite and 5 to 10 grams gold per ton; the other yields ore consisting mostly of schistose material and some quartz,

containing 5 per cent pyrite and 4 to 5 grams of gold. Chemically the ore was ideal for cyanidation, but physically the schistose matter complicated thickening and filtration. Treatment consisted of grinding to 35 mesh, amalgamating, separating the pulp into sand and slime, recovering the mineral on tables and cyaniding the three products. Although 60 per cent of the gold is free, the mercury saved but 35 per cent of it. The schist in the ore reduced the mill capacity to a third by preventing proper and prompt settling of the pulp and by clogging the cloth on the Oliver filters. Sorting out some of the troublesome gangue was tried, but as a third of the ore by-passes the sorting section as fines, which carry a high proportion of schist, a clean quartz feed is impossible. Eventually, after much experimentation in the laboratory and in a pilot mill of 3 tons capacity, the flow-sheet decided upon was grinding, amalgamating, floating all of the pulp and cyaniding only the concentrate. Concentration was 35 into 1; the reject formed 97 per cent, and the concentrate 3 per cent of the pulp. This simplified process gives an extraction equal to all-cyanidation at a lower cost.

There are also mills in which flotation equipment was put in for the purpose of recovering the gold in a concentrate and rejecting the tailings only to find that the tailing was too valuable to waste and must be cyanided before discarding.

It is generally true that cyanidation is capable of producing a tailing of lower gold content than flotation. At a price of \$35 per oz. for gold this fact is of much greater importance than when gold was valued at \$20.67 per oz. This value of the residue to be discarded will influence the choice of a method of treatment.

Flotation as an Accessory.—In his review of the prevailing practice in the flotation of gold in *M.M.*, September, 1934, Philip Rabone points out that treatment of ores carrying gold and silver associated with pyrite and other iron sulphides, arsenopyrite or stibnite, depends on how the gold and silver are associated with the sulphides. Cyanidation is usually the most suitable process, but it often necessitates grinding ore to a fine size to release the gold and silver. Should it be possible to obtain a good recovery by flotation in a concentrate carrying most of the pyrite or other sulphides, it might be more economical to adopt this method, regrinding only the comparatively

small bulk of concentrate prior to the leaching operation. Flotation may also prove to be the more economical process for the ore containing such minerals as stibnite, copper-bearing sulphides, tellurides and others, which require roasting before cyanidation, because this reduces the tonnage passing through the furnace. Even when recovery of gold and silver from such ores by flotation is low, it may be advantageous still to float off the minerals that interfere with cyanidation, roasting and leaching, or possibly smelting the concentrate for extraction of its precious metals. Cyanidation of the flotation tailing follows, this being simpler and cheaper because of prior removal of the cyanicides.

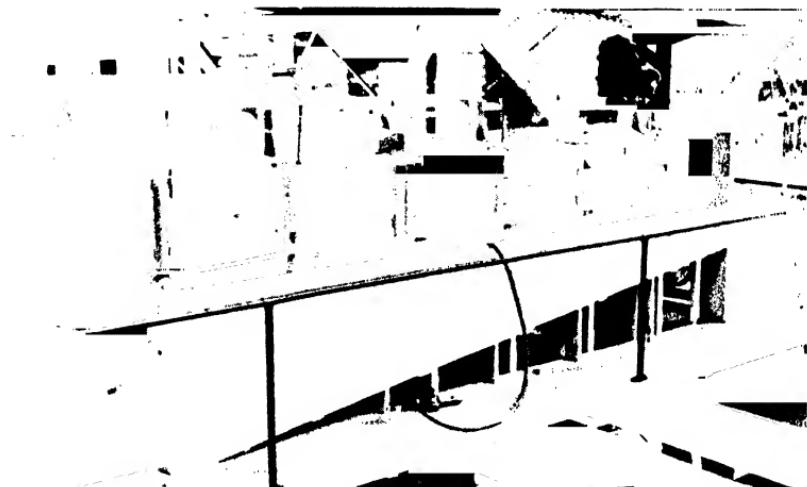


FIG. 57.—Kraut flotation cells on Bulolo dredge 3.

Flotation Unit in the Grinding Circuit.—It is good practice to recover as much of the gold and silver as possible in the grinding circuit by amalgamation or on corduroy strakes to prevent their accumulation in the classifier; otherwise an occasional rush of material from the grinding section may carry gold that is too coarse to float into the flotation circuit and will thus pass into the tailing and be lost.

To prevent this, several companies, including the Golden Cycle at Colorado Springs and the McIntyre-Porcupine at Timmins, Ont. have inserted a combination of flotation cells and hydraulic cone in their tube-mill classifier circuits. At the McIntyre-Porcupine, according to J. J. Denny in *E. and M.J.*,

November, 1933, this cell is a Denver 500 Sub-A-type. The total pulp discharged from each tube mill passes through 4-mesh screens which are attached to the end of the mills. The undersize goes to the flotation cell, and the oversize to the classifiers. Tailing from the cell flows to the classifiers, and the concentrate to a collecting box. The purpose of the hydraulic attachment is to remove gold that is too coarse to float, thus avoiding an accumulation in the tube-mill circuit. The cones have increased recovery from 60 to 75 per cent. Every 24 hr. the tube-mill discharge is diverted to the classifiers. Water is added for 15 min. to separate the gangue in the cells from the high-grade concentrate, after which a product consisting of sulphides and coarse gold is removed through a 4-in. plug valve equipped with a locking device. Each day 400 lb. of material worth \$3000 is recovered. This is transferred to a tube mill in the cyanide circuit, with no evident increase in the value of the cyanide residue.

As proof of the value of these gold-flotation cells, two comparative runs were made during 1933, the tube mill and classifier circuit being with and without flotation. Results were as follows:

EFFECT OF GOLD-FLOTATION CELL IN TUBE-MILL CLASSIFICATION CIRCUIT

Item	Test 1		Test 2	
	With cell	Without cell	With cell	Without cell
Tube-mill feed (ore), tons.....	425	425	425	425
Value per ton ^a	\$7.64	\$7.64	\$7.43	\$7.43
Tube-mill discharge value ^b	\$14.05	\$35.21	\$26.34	\$44.06
Concentrate value.....	\$126.91	\$100.32
Classifier rake return value.....	\$30.59	\$83.32	\$28.80	\$48.34
Classifier overflow value.....	\$3.29	\$7.45	\$2.84	\$7.24
Recovery in concentrates, per cent.....	58.45	63.54

^a Original ore.

^b Includes classifier sand return.

Gold at \$20.67 per oz.

Possibilities on the Rand.—On the Rand, the major portion of the residual gold in the ore after thorough cyanidation is totally encased in the pyrite, which constitutes only about 2 per cent of

the total weight of the ore. Stage and selective grinding of the sulphides is now practiced. Flotation had been tried prior to and during 1924 at Modder Deep to separate a pyritic concentrate for selective grinding. Flotation is again being seriously considered for the treatment of Rand ores, according to T. K. Prentice in *Bul., I.M. and M.*, April, 1935. Experiments have been made at the Simmer and Jack mine and at the Government Areas. At both of these plants the sand residue is relatively high in gold, and it is probable that a large plant will be built at each of these mines to separate a flotation concentrate from the original sand. The concentrate will be reground and cyanided, and the flotation residue will pass on to the existing sand plants for cyanide treatment.

CONCENTRATE TREATMENT

In the treatment and disposal of gravity and flotation concentrates both metallurgical and economic factors must be considered. In general, local treatment is preferable, but proximity to a smelting plant, low transportation rates and reasonable smelting charges may favor the shipping of concentrates over making the capital outlay for a local plant and treatment.

For local treatment these metallurgical questions must also be answered: Shall the concentrate be treated raw or roasted? Shall the concentrate residue be discarded or returned to the main pulp flow for further treatment?

The prevailing practice in Australia is to treat concentrates by roasting and cyaniding; in North America and on the Rand treatment of the raw concentrate is preferred, but roasting is being considered in some mills.

Ernest Gayford, in *Trans. 112, A.I.M.E.*, 1934, in discussing ore treatment as a factor in small gold-mining enterprises (15 to 150 tons per day), at the higher price (\$35. per oz.) for gold, concludes that, as with base metals, the nearer the producer can get to making a finished product himself the greater will be his reward, the net returns for bullion being necessarily higher than for the same amount of gold sent to a smelter in concentrates. On a shipment of bullion containing 100 oz. fine gold, the return would be \$34.81 an ounce. If 50 tons of 2-oz. concentrate, 20 tons of 5-oz. concentrate or 20 tons of 10-oz. concentrate were

sent to a smelter, the net respective yields based on average conditions would be \$26.03, \$28.94 and \$30.64 per oz. Of course the charges against the concentrates include haulage, freight and treatment.

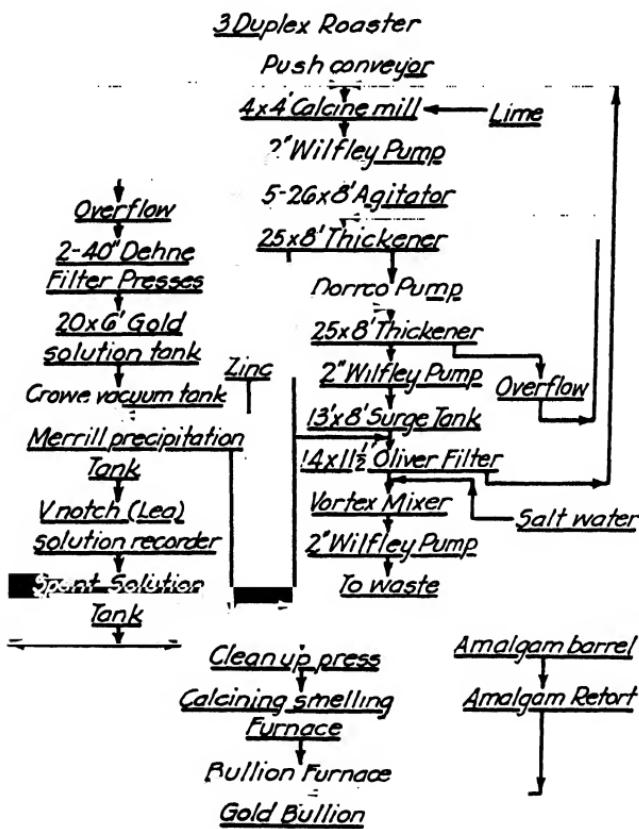


FIG. 58.—Flow-sheet, concentrate treatment, Lake View and Star, Kalgoorlie.

Idaho-Maryland Concentrate Treatment.—About 70 to 80 per cent of the gold in the ores of the Idaho-Maryland Mines Corporation occurs in the free state, the remainder being associated with sulphides.

About 90 per cent of the free gold is recovered on Hungarian riffles, Deister tables or by plate amalgamation. Some of the sulphides are taken off the cleaner tables, and the remainder are floated from the table tails. In addition, the flotation plant is

treating old tailings, and the combined concentrate goes to the cyanide plant.

The concentrates are cyanided in a plant having a capacity of about 30 tons per day. Figure 59 gives the flow-sheet of this plant. Concentrates after filtering are weighed and sampled and then reground in cyanide solution in a 5 by 18-ft. Allis-

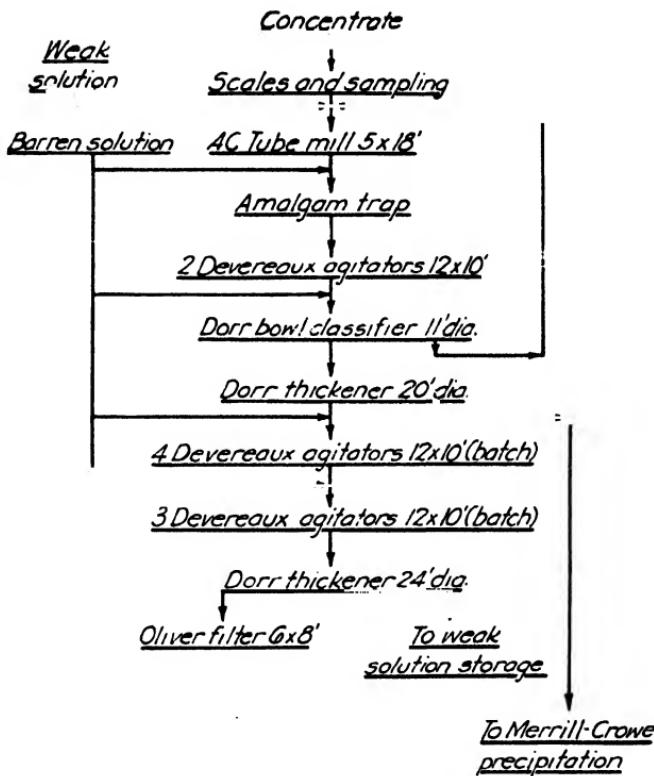


FIG. 59.—Flow-sheet of Idaho-Maryland concentrate plant.

Chalmers tube mill in closed circuit with two 12- by 16-ft. Devereux agitators and an 11-ft. diameter Dorr bowl classifier. No. 2. Danish pebbles are used in the mill.

The bowl overflow is held at 12 to 15 to 1 dilution at which the maximum size of sulphide particles overflowing is 400 mesh. The bowl overflow goes to a 20-ft. diameter Dorr thickener. The overflow is precipitated and the underflow at 60 to 75 per cent solids goes to one of four 12- by 10-ft. Devereux agitators. The

pulp is diluted to 3 to 1 dilution with barren solution and agitated 36 hr. The entire charge is then pumped to one of three 13- by 12-ft. Devereux agitators, more barren solution is added and 48 hr. additional agitation given. A 24-ft. Dorr thickener dewateres the pulp, the overflow goes to a weak-solution tank and the underflow is filtered on a 6- by 8-ft. Oliver filter. A barren-solution wash is used on the filter.

Merrill-Crowe simultaneous clarification and precipitation is used. Two men are required to operate the plant at full capacity.

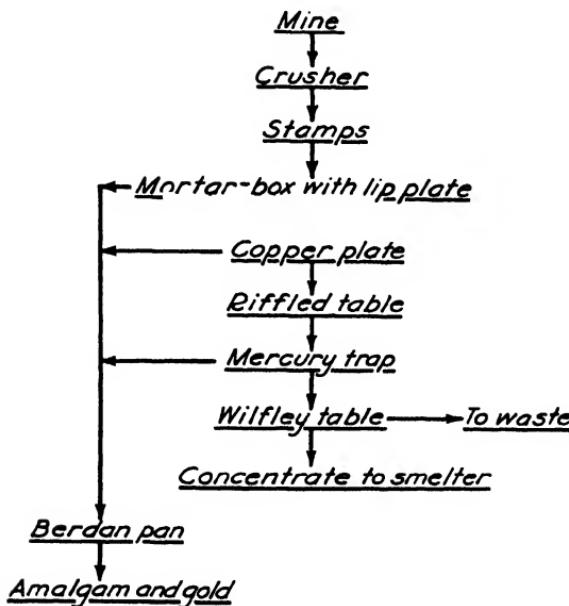


FIG. 60.—Forty-ton amalgamating and concentrating plant.

Reagent consumption averages: Cyanide, 7 lb.; lime, 10 lb.; flint pebbles, 5 lb. per ton concentrate.

Willow Creek Treatment.—From 40 tons of ore (see Fig. 60) the Willow Creek Mines Company, Luckyshot, Alaska, collects 1500 lb. of concentrate daily. This consists of pyrite and arsenopyrite, with small amounts of chalcopyrite and azurite. Some of it is saved on a Gibson table, and some in a minerals-separation flotation machine. The high-grade table concentrate runs through a batea, the tailing of which together with the low-grade table concentrate and the flotation concentrate goes to an

amalgamating pan operating in closed circuit with a Hardinge mill and Akins classifier. Grinding is done in lime water to 98 per cent through 200 mesh. Pumping is done by air lift to provide additional aeration with the lime solution prior to cyanidation.

Cyanidation consists of treating batches of 3000 lb. of concentrate by intermittent agitation, settling and decanting. This is done in three 6- by 8-ft. Devereux agitators which have an 18-in. central downcast pipe. Air at 25-lb. pressure is used. Of the 140 hr. of sulphide-cyanide contact, 64 hr. is by agitation. The consumption by cyanide is 15 lb., and of lime 25 lb. per ton of concentrate. The loss of dissolved gold is \$1 per ton. Zinc shavings, 8 lb. per ton of concentrate, precipitate the gold from a pregnant solution assaying \$30 per ton. One-tenth pound of lead acetate per ton of solution is added to the clarifying sand filter.

Recovery of the original gold in the ore totals 96 per cent--60 per cent in the batea, 20 per cent in the pan and 16 per cent from the concentrates.

Shipping to Smelter.—Read, Denny and Hutchison, quoted later on treatment of silver ore, give schedules of the cost of sending concentrates to a smelter. They considered that a most serious objection to the flotation process is the difficulty of treating the concentrates to recover the silver as fine metal. In general, the freight from Cobalt, Ont., was \$24.50 per ton; smelter treatment charge, \$14. per ton; smelter deduction, 5 per cent; refining charge 1/4 ct. per oz.; representation and assaying, 86 ets. per ton. On six grades of concentrates the charges and returns were as follows, silver then being 65 cts. per oz.:

COST OF AND RETURNS FROM SMELTING CONCENTRATE

Grade, ounces per ton	Value per ton	Charges per ton	Net per ton	Yield, per cent
100	\$65.00	\$42.76	\$22.24	34.21
200	130.00	46.26	83.74	64.41
300	195.00	49.76	145.24	74.48
400	260.00	53.26	206.74	79.51
500	325.00	56.76	268.24	82.53
600	390.00	60.26	329.74	84.55

These figures show a marked advantage in producing and shipping a high-grade concentrate.

At the California Rand Company mill in Southern California the flotation concentrate was shipped to a smelter with rich crude ore. During 1932 the results were as follows:

Concentrate shipped, tons.....	6,918
Value.....	\$1,747,702
Ores shipped, tons.....	5,013
Value.....	525,821
Smelter charges:	
Treatment.....	147,851
Deduction, 5 per cent.....	104,279
Antimony penalty.....	9,388
Freight.....	113,219
Total.....	\$374,737

Although the charges against the concentrates were not shown separately, the average for 11,931 tons of concentrate and ore was \$31 per ton. The penalty for antimony was almost 80 cts. per ton. However, smelting probably was the easiest way to handle the concentrate, especially as crude ore was being shipped.

RAW TREATMENT

Raw treatment of concentrates usually requires fine grinding and prolonged contact—up to 16 days—with cyanide solutions. Concentrates in which the gold is associated with pyrite only usually respond satisfactorily to raw treatment by cyaniding following fine grinding; when the gold is associated with other minerals, raw treatment frequently does not give satisfactory extraction.

NORTHERN ONTARIO TREATMENT

Practice in "handling" sulphides in the Kirkland Lake and Porcupine districts varies from grinding and treating them concurrently with the remainder of the ore to entirely separate treatment and is summarized as follows for the plants visited:

Kirkland Lake District.—Teck-Hughes and Toburn grind the ore as a whole and cyanide it by agitation and filtration and by decantation and filtration, respectively.

Lake Shore floats its sulphides and tellurides after the whole pulp has been cyanided, then regrinds the minerals and cyanides them again. Separate precipitation circuits are maintained for the solutions from the regular mill and the concentrate-treatment plant.

Wright-Hargreaves floats its sulphides after the whole pulp has been cyanided, regrinds the minerals and returns them to the

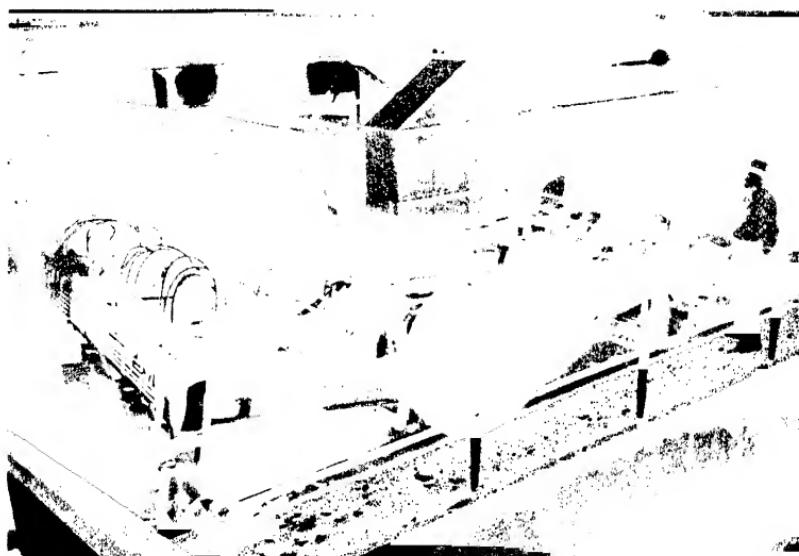


FIG. 61.—Twelve-foot-wide Dorr FX classifier in closed circuit with 9- by 7-ft. Allis-Chalmers ball mill at Wright-Hargreaves, illustrating raising of ball-mill discharge to classifier by means of spiral-scoop elevator, and gravity return of classifier sands.

circulating pulp for further treatment. There is no accumulation of sulphides in the circuit.

Porcupine District.—*Coniaurum* grinds the ore as a whole and cyanides it by agitation and filtration.

Dome, by means of cone classifiers, builds up the sulphides in the pulp from 3.75 to 9 per cent. These are ground separately, passed over corduroy tables and mixed with the remainder of the pulp for agitation and filter pressing.

Hollinger collects its sulphides on Deister double-deck tables, regrinds them and then mixes them with the remainder of the pulp for agitation and filtration.

McIntyre-Porcupine floats its sulphides and cyanides them only. The remainder of the pulp is discharged to waste.

ROASTING TREATMENT

The roasting of concentrates is old practice and has been used in many parts of the world preceding chlorination, cyanidation or shipment to smelters.

EXAMPLES OF ROASTING

Cam and Motor Treatment.—Some interesting analyses of ore and concentrates at the Cam and Motor mine, Rhodesia, are given by Digby Burnett, in *M.M.*, February, 1926:

ANALYSES OF CAM AND MOTOR ORE AND CONCENTRATES, PER CENT

Constituent	Ore	Vanner concen- trates	Flotation concen- trates	Roasted concentrates
Silica.....	58.77	49.97	28.93	53.25
Arsenic.....	0.51	2.86	4.09	0.60 (oxide)
Antimony.....	1.19	1.36	7.80	2.57 (oxide)
Sulphur.....	2.13	7.63	15.70	5.12 (SO_2)
Iron.....	4.50	9.80	14.60	14.30 (oxide)
Alumina.....	7.35	7.79	10.73	9.86
Lime.....	6.16	5.33	3.38	5.73
Magnesia.....	4.32	3.89	2.51	3.00
Carbon dioxide.....	8.58	7.77	5.30	3.46
Titanium oxide.....	0.20	0.17	Trace	0.20
Graphite.....	Trace			
Manganese.....	Trace	Trace	Trace	Trace
Water.....	5.17	3.10	5.82	2.35
Total.....	98.88	99.67	98.86	99.94

The treatment of the concentrates is not simple; by roasting and cyanidation an extraction of 90 per cent is made.

In a recent communication, Mr. Burnett, general manager and consulting engineer, reports that prior to 1926 flotation concentrates were mixed with gravity concentrates and sent to the roasters. Between July, 1926, and April, 1931, because of the high percentage of antimony in the ore, the flotation concentrates were sent to a blast furnace. In April, 1931, in consequence of a considerable decrease in the antimony content, the blast furnace

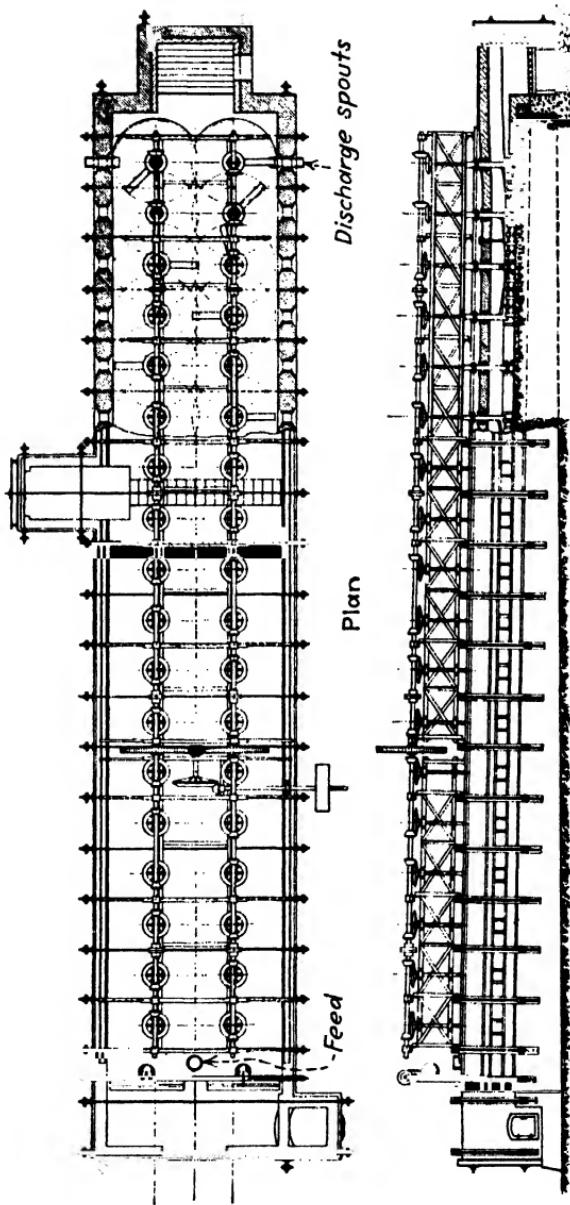


Fig. 62.— Plan and side elevation of Edwards duplex roasting furnace.

and flotation plant were shut down, and 34 James slime tables were installed. All concentrates from these tables are mixed with those from the Record vanners and are sent to five Edwards roasters (see Figs. 62 and 63).

Flue dust from these furnaces is agitated with cyanide and filter pressed, the residue being pulped, dewatered and fed to the roasters for further treatment. Roasted ore is first cyanided, weathered for four months, ground with cyanide, run over blankets, separated into sand and slime in a cone and Dorr classifier and the two products cyanided. Gold is precipitated on zinc shavings.

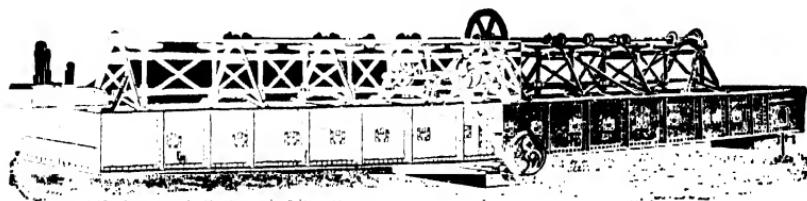


FIG. 63.—Perspective view of Edwards simplex adjustable furnace.

Golden Cycle Treatment.—As explained elsewhere, the Golden Cycle mill at Colorado Springs treats the products from three different sections of the plant. These are roasted in eight Edwards furnaces—one for coarse and seven for fine material. The coarse material contains 96 per cent plus 8 mesh; the other feed is from plus 8 to minus 200 mesh. This sizing reduces fuel consumption. Each furnace will roast 115 to 130 tons per day.

The duplex furnaces (see Fig. 62) have a hearth area of 13 by 115 ft. with a slope of 2 in 12 in. Each furnace has 27 water-cooled rabbles in each line traveling at 3 r.p.m. The ore passes through the furnace in 6 hr. Each furnace requires 16 hp. At the discharge end is a cooling hearth 13 by 44 ft.; the rabbles in this section make 6 r.p.m. Calcines from the cooling hearth are moved by a 350-ft. drag conveyor which has sheet-iron fins or flights 8 in. deep and 34 in. long. The conveyor consumes as much power as one furnace. At the end the ore is cooled to 85°C. The roasted ore is finally ground and cyanided.

Colorado lignite of a heating value of 8700 B.t.u. is burned in a semigas producer type of firebox using live steam. The consump-

tion of coal is 230 lb. per ton of ore roasted, plus 8 per cent to generate the steam. By care in feeding the charge into the furnaces, by better draft control and by choking the calcine discharge, the metal loss in stock dust has been reduced to 4 cts. per ton of ore. Indicating and recording pyrometers are used, and sulphur analyses are made daily.

The formation of calcium sulphide and calcium sulphate from basic ores is undesirable, the former decreasing dissolution of the gold and precipitating silver, under certain conditions. The sulphate shows its presence at various points in the equipment. Therefore, careful attention to the roast is essential.

Arsenic is also recovered from the flue gas from the concentrate-roasting furnaces at the Wiluna gold mine, Western Australia. The flotation plant there treats 1500 tons per day.

ACID TREATMENT

The treatment of material by acid in order to prepare it for the more economical application of cyanide or other metallurgical means of gold or silver recovery has been suggested in many cases, and experiments have been made to determine the possibility of using it. In only a few cases, however, has it been tried on a practical working scale, mainly in preparing accumulated panamalgamation tailings for cyanidation. Results have not been encouraging, since it has been found extremely difficult, if not impossible, to remove the acid so completely as to avoid high consumption of cyanide in the subsequent treatment.

CHAPTER X

BULLION RECOVERY

Regardless of subsequent treatment in an ore, it is best to recover the free gold as early and as completely as possible in the flow of pulp. For this purpose there are available and suitable mercury in stamp mortar boxes if they are used, amalgamated copper plates, various traps, tables, corduroy, stakes, jigs or a unit flotation cell and hydraulic cone in the grinding-classifying circuit. Information is given regarding the use of amalgamation following stamps and following a ball mill and on the use of corduroy stakes which is supplanting copper plates. The hydraulic trap and the unit flotation cell are being used in increasing numbers. These devices and their application are described.

Precipitation of gold and silver on zinc dust or on zinc shavings is fully treated, from the clarification and deaeration of cyanide solutions to the cleanup and melting of the bullion. The use of aluminum dust, sodium sulphide and charcoal as precipitants receives attention.

THE UNIT FLOTATION CELL AND HYDRAULIC CONE

To save coarse gold and minerals at an early stage in milling some plants are installing the Denver Equipment Sub-A flotation cell and hydraulic cone. Figure 64 shows its place in the circuit at the McIntyre-Porcupine mine, Ontario. The Golden Cycle mill at Colorado Springs uses a unit cell between ball mill and classifier and treats the rich concentrate in a small Wheeler-type pan. The flotation machine and its use in this circuit have been patented in several countries.

The unit flotation cell prevents the accumulation of free gold in a ball-mill classifier circuit by removing coarse gold as soon as possible. Otherwise, because of its high specific gravity, the gold remains in this circuit until finely ground. Also, periodic surges of the classifier may allow this accumulation of gold to get into the mill circuit with resulting loss because the regular treatment is not effective on the high-grade pulp. Moreover, the gold sufficiently fine to overflow the classifier lip is likely to become coated with sulphide slime during the grinding operation, reducing the recovery by flotation.

HYDRAULIC GOLD TRAP

The hydraulic trap, another method of catching gold, is described by Ernest Gayford in *Trans. 112, A.I.M.E.*, 1934, and is shown in Fig. 65. It is a simple device inserted in the mill-classifier circuit; sometimes two or more are used in series. At the Montezuma-Apex plant, Nashville, Calif., the 65-mesh



FIG. 64.—Denver Equipment Company unit flotation cell and hydraulic cone in tube-mill classifier circuit.

discharge from each of two Marcy mills passes through five hydraulic traps, followed by corduroy tables, Dorr classifiers and flotation. The traps are bled daily and thoroughly cleaned out twice a month. The product is concentrated periodically on a small Wilfley table. The concentrate is ground and amalgamated in a barrel, and the tailing returned to the mill circuit.

AMALGAMATION

Gold recovery by mercury, or amalgamation, which has saved so much of the gold of the world, is, at large mines, becoming

of less importance. Probably at small operations, amalgamation will be continued, unless corduroy supplants it. Because the methods are generally known, brief descriptions only are given of amalgamation in the mortar boxes of stamps and on copper

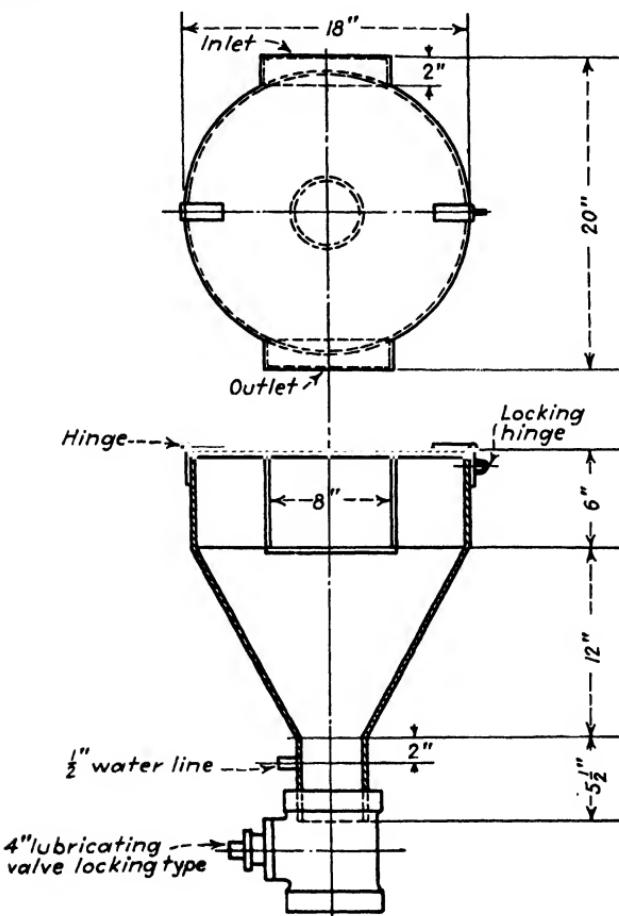


FIG. 65. Hydraulic gold trap.

plates and the handling of amalgam. Figures 59 and 60 show flow-sheets of amalgamating plants.

A Berdan pan is effective for cleaning mercury or amalgam or for amalgamating corduroy gold. The pan of this type, made by the Mine and Smelter Supply Company, is suspended at an angle from a guarded, gear-driven spindle within a frame of

angle iron standing $4\frac{3}{4}$ ft. high. The bowl or pan is 24 in. in diameter and runs at 20 r.p.m., taking $\frac{3}{4}$ hp. Grinding is done by means of an 8-in. steel ball running loose in the bowl. Material as coarse as $\frac{1}{4}$ in. may be fed to the pan. A Berdan pan can be fed continuously or intermittently. If the latter method is employed, when the material has been ground, a supply of water will wash out the slime and leave clean amalgam. This pan can be used for the cleanup or for grinding concentrates at a small mine or for grinding and amalgamating rich ore.

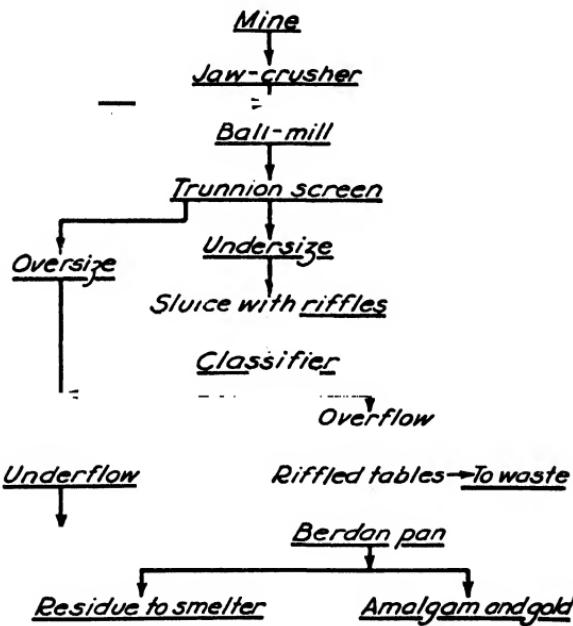


FIG. 66.—One-hundred-ton mill, 90 per cent of the gold caught on riffles.

Amalgamation at Alaska-Juneau.—A concentrate assaying \$20,000 to \$30,000 in gold per ton collected from concentrating tables each day is ground in a 2- by 4-ft. barrel, running at 30 r.p.m.; a piece of 6-in. shafting is the grinding medium. To the concentrate are added 600 oz. mercury and 3 lb. freshly slaked lime, and the mixture is ground for 5 or 6 hr. The barrel is discharged, while running, through a $1\frac{1}{2}$ -in. hole into a 4-ft. mechanical batea, which removes most of the galena. Residue in the batea is cleaned by hand, and the amalgam squeezed in an

air-operated press. The concentrate discharge from the batea averages \$75 gold per ton.

The amalgam is retorted in an oil-fired, cast-iron, horizontal, cylindrical retort, in which are placed small, semicircular, cast-iron boats. The amalgam yields 60 to 65 per cent gold, and the mercury loss is 1 per cent. This gold is melted in an oil-fired tilting furnace, soda and borax glass being used as flux. The molten metal is sampled with a graphite sampler. Forty minutes is required to melt 1200 to 1600 oz., which makes one bar. The bullion assays 850 fine in gold and 130 fine in silver.

One man takes care of amalgamating, retorting and melting.

Amalgamation at Argonaut, Calif.—Mortar-box and plate amalgamation at the Argonaut mine, Mother Lode district, saves 70 per cent of the gold, according to S. E. Woodworth in *I.C. 6476, U.S.B. of M.*, 1931.

Amalgamation is practiced both inside and outside the 12-stamp batteries. For inside amalgamation, straight-sided, copper-faced, wooden chuck blocks are used. These blocks are equipped with half-round iron strips, spaced 2 in. apart, for their entire length. It was found that the half-round strips assisted amalgam to build up to a greater thickness than did the smoother block. If the hourly inspection of the chuck blocks indicates that mercury should be added, it is fed in measured quantities from a horn spoon with the incoming ore at the back of the batteries. The chuck blocks are cleaned twice a month or oftener. The total quicksilver fed is recorded and gives a close estimate of the free-gold content of the ore crushed.

The pulp passing the battery screen falls upon reverse splash plates, the first of which is 5 by 51 in. in size with an area of 1.77 sq. ft., set at a slope of 3 in. per ft., and the second is 8 by 51 in. in size with an area of 2.83 sq. ft. and an inclination of 4 in. per ft. There is a drop of 2 in. between these plates. On leaving the reverse splash plates the pulp drops 5 in. to the cast-iron lip of the mortar. This drop is variable, depending on the height of the battery discharge. From the mortar lip, the pulp falls 4 in. to an amalgamated apron plate, set at a slope of 1 $\frac{1}{4}$ in. per ft. This plate is 49 by 58 in. in size and has an area of 19.4 sq. ft. An amalgam trap consisting of a wooden box of length equal to the width of the plate and of a uniform depth of 8 in. is attached to and forms a part of the apron frame. The pulp flows from this trap through four 2-in. iron nipples, set in the side of the box, on a plane 4 in. from the bottom. The pulp, issuing from these nipples, drops through a 10-mesh, woven-wire, brass screen to the sluice plate. If a battery screen is punctured, this 10-mesh screen catches the coarse oversize.

The total drop from the discharge nipples of the trap to the sluice plate is 6 in. This plate is set at an inclination of $1\frac{1}{4}$ in. per ft. and is $46\frac{1}{2}$ in. by 16 ft. in size. It has an area of 62.10 sq. ft. The total length of plates per battery is about 21 ft., and the total plate area is 86 sq. ft. On the average tonnage this represents about 4.3 sq. ft. of plate area per ton of ore milled per day. As the pulp leaves the sluice plate it drops into another amalgam trap which is attached to and forms a part of the plate frame. This trap is a wooden box built the entire width of the plate, but unlike the first trap it has a sloping bottom; and the depth of sand adjacent to the plate discharge is but 4 in., whereas the depth along the trap overflow is 6 in. All plates, including the chuck blocks, are made of copper which weighs 5 lb. per sq. ft. and are electroplated with 3 oz. of silver per sq. ft. of contact surface.

When the tube mills operate and grind the sands from the classifier, which treats the tailings of the first two rows of vanners, the ground product is amalgamated on eight shaking plates 4 by 5 ft. in size, which have a combined area of 160 sq. ft. These plates are set upon a slope of $3\frac{1}{2}$ in. per ft. and are oscillated ninety times a minute by a simple strap eccentric.

Each morning the amalgam on the apron and sluice plates is softened with mercury, rubbed with a rag, cleaned with a rubber squeegee and dressed with a whisk broom for the day's operation. The cleaning and conditioning of these plates require about $3\frac{1}{2}$ hr. The monthly cleanup, including retorting of amalgam and melting of bullion, occupies 5 days. On the first day of the cleanup all sluice plates are cleaned and scraped with wide-faced putty knives, and in addition one battery is dismantled, cleaned and reassembled. During the second day six apron plates are cleaned and scraped with a scraper made from an old file, the end of which has been flattened, widened, sharpened and turned at right angles to its length. Two additional batteries are cleaned on this day. On the third day, the remaining six apron plates are scraped and cleaned, and three more batteries are cleaned out. During the fourth day all splash plates are taken to the cleanup room to be steamed and scraped, and, in addition, four batteries are cleaned up. The two remaining batteries are cleaned on the fifth day.

The amalgam recovered from dressing the plates each day is squeezed into a pellet and stored until retorted with the general cleanup amalgam. Residues taken from the batteries are placed in an amalgam barrel, which contains three pieces of stamp stem and which is rotated for about 12 hr. Mercury, amounting to 350 troy oz., is then placed in the barrel, which is again rotated for an additional hour or two. It is then stopped, opened, cleaned and the pulp run into a storage box under the barrel. From here it is fed by the cleanup man to a power jig, the bed of which forms on a fine wire screen. The amalgam is found in the

hutch of the jig with the iron floating on the amalgam, and the sand over the iron. When all of the barrel charge has passed through the jig, the sand is scooped from the hutch, the iron is removed by a magnet and the liquid amalgam is removed through a spigot into an iron dipper. Here it is further cleaned by mechanical agitation and a water jet which removes any foreign matter present. After squeezing the cleaned amalgam in a canvas cloth by hand and removing most of the liquid mercury, additional mercury is removed by further squeezing the soft amalgam in canvas, using a mold and a hydraulic press.

The final amalgam from the cleanup, placed in trays, is sealed in the retorts during the afternoon of the fifth day. The retorts are heated, and the quicksilver volatilized and condensed during the night. Wood is used for fuel. The next morning, the sixth day, the bullion sponge is removed from the warm retort, placed in graphite pots and, after melting in oil-fired furnaces, is poured in bars which are shipped at once either to the American Smelting and Refining Company at Selby or to the United States mint at San Francisco. The trays into which the amalgam is placed for retorting are painted with a chalk-and-water emulsion which is thoroughly dried before using. This coating assists in a clean removal of the gold sponge after retorting.

During 15 years the bullion has averaged 821 parts gold, 159 parts silver, and 20 parts base metal. The mercury loss is 0.17 troy oz. per ton of ore milled.

Amalgamating Ball-mill Product.—At one time it was thought by some millmen that only the pulp from stamps was suitable for amalgamation on plates. That this is not so today is shown by the practice at the Porcupine United plant at Timmins, Ontario, in which plate amalgamation follows ball milling. The ore consists of quartz with fine stringers of schist or of banded quartz schist or of stringers of quartz in schist. About 75 per cent of the gold is free and fairly coarse, according to R. A. Vary in *I.C. 6433, U.S.B. of M.*, 1931. Half-inch ore is ground to 65 per cent through 200 mesh in a $4\frac{1}{2}$ -ft. by 16-in. Hardinge mill. Amalgamation, as indicated in Fig. 67, is done as follows:

The ball-mill discharge falls into a distributing box where water is added, and the flow distributed to two 4- by 8-ft. copper plates $\frac{3}{8}$ in. thick and with a slope of $1\frac{5}{8}$ in. per ft. The box traps considerable coarse gold. It is cleaned out once a week, and its contents run through an amalgamation barrel. The table frames supporting the amalgamation plates are constructed of 2- by 4-in. lumber, with crosspieces placed on edge and spaced 6 in. apart. The table decks are made of 1-in.

shiplap on which copper plates are screwed. The slope of the deck can be changed to suit conditions by placing wedges between the deck and the table frame. The plates are not silvered, and the copper has to be treated with a weak cyanide solution before the mercury will amalgamate with the copper. Little trouble has been experienced by copper showing on the plates.

Mercury is shaken on the top half of the plates, and none is added elsewhere. Ordinarily the plates are dressed every 3 hr., but oftener if

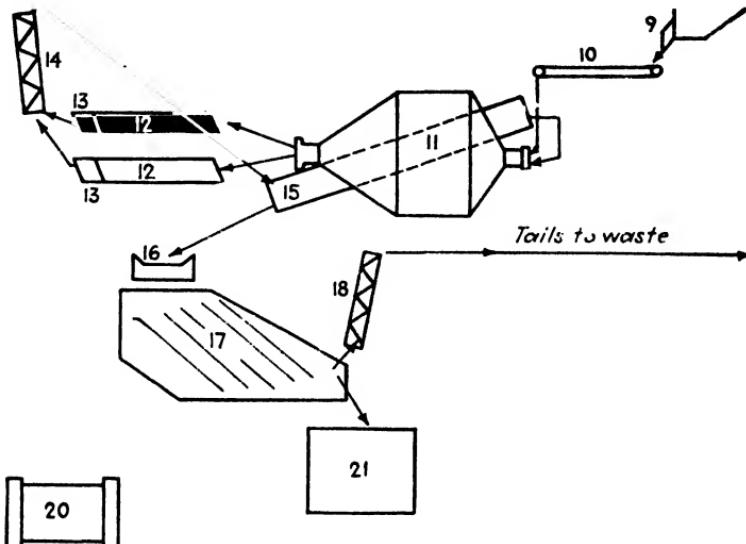


FIG. 67. That part of the Porcupine United mill showing amalgamation practice.

9. Chute.	16. Gibson amalgamator.
10. Ball-mill feed conveyor.	17. James tables.
11. Ball mill.	18. Tailings elevator.
12. Amalgamation plates.	19. Water tank.
13. Blankets.	20. Amalgamation barrel.
14. Bucket elevator.	21. Concentrates bin.
15. Dorr classifier.	

the ore is rich. The method employed in cleaning a plate is to by-pass all the feed to the other plate, clean off all ore particles, then brush the plate well with a stiff whisk broom, working any loose amalgam to the top of the plate. The loosened amalgam is removed, and if this leaves the plate too dry, mercury is shaken on and rubbed in well. The plate is then brushed horizontally, working from the center to the sides and starting at the bottom and working to the top of the plate. Any amalgam or loose mercury adhering to the sides of the plate is then brushed to the top or removed if the amount is appreciable.

On the morning shift the plates are given an extra brushing, and mercury is added to loosen the amalgam. Then the amalgam is stripped

off with a piece of rubber conveyor belting, stripping being done at right angles to the slope of the plate; the amalgam is lifted and the plates redressed in the usual manner. Care is taken that the plates are not stripped too clean.

Little crystallization of the copper takes place, and the plates are rubbed occasionally with a weak cyanide solution, which removes any tarnish or stains. Enough water is used to maintain an even flow of pulp over the plates, and when the ball-mill discharges too much coarse material, the feed is cut off for a short period. Forty per cent of the total gold recovery is made on the plates.

On the lower end of each table below the plate is fastened a sheet of $\frac{1}{4}$ -in. iron plate, 42 by 48 in. A blanket of No. 6 silence cloth is laid on this plate and secured by a flat iron bar $\frac{1}{8}$ in. thick and 2 in. wide laid on top of the blanket. The bar is held by notches cut in the table frame. The pulp from the amalgamation plates passes over the blankets; pyrite, fine gold and mercury from the plates are caught on the blankets. The blankets are changed and washed in a tub after each dressing of the plates, the blanket concentrates being sent to an amalgamation barrel for treatment. The plate and blanket tailings drop to amalgam traps at the end of each table and are elevated therefrom by a bucket elevator which returns them to the Dorr classifier. This elevator has a deep sump which is a good trap. The sump is cleaned out at regular intervals. Amalgamation takes place in the whole circuit. Elevator discharge launders, classifier and other launders all collect rich sand and amalgam; these are cleaned out periodically and treated in the amalgamation barrel. Thirty-five per cent of the gold is recovered by barrel treatment.

The rake product from the Dorr classifier is returned to the ball mill, and the overflow runs by gravity to a Gibson impact amalgamator, attached to a James sand-concentrating table. The amalgamator catches float mercury and fine gold which has escaped the amalgamation plates, blankets and traps. The amalgamator is opened and washed, and its plates are scraped once a week. The James table is operated at 250 r.p.m. with a $\frac{3}{4}$ -in. stroke. The table concentrates, averaging \$40 per ton, are dewatered and sent for cyanide treatment to an affiliated company. The table tailings are elevated by a bucket elevator to the tailings dump. The table concentrates and table tailings are both sampled hourly by hand by taking dip samples from their launders.

The concentrates or sands from the blankets, traps and launders are ground for 10 hr. in a cast-iron amalgamation barrel, 16 in. in diameter and 36 in. long, revolving at 22 r.p.m., using worn balls from the ball mill as grinding media. Then about 250 oz. of mercury and 3 lb. of slaked lime are added to the charge, and it is again ground for 5 to 8 hr. The barrel is washed out into a box, the iron balls are carefully cleaned

by hand, and the residue is run over a small amalgamation plate to the mill circuit. The mercury and amalgam are collected, washed and cleaned with hot water and then squeezed by hand through fine sheeting to eliminate excess mercury, retaining the amalgam in the form of a ball.

The amalgam is retorted outside the mill over a wood fire at regular intervals, using a cast-iron retort which has a capacity of 1000 oz. The sponge-gold recovery is 35 to 40 per cent the weight of the amalgam retorted, and the mercury loss is small. The sponge gold is melted in an oil-burning furnace at the affiliated company's refinery. Soda, borax and manganese dioxide are used for flux, and the molds are coated with lampblack. The bullion is sampled by drilling small holes in opposite ends of the bar at top and bottom. The average grade of bullion is 770 fine in gold and 120 to 140 fine in silver.

The crew for the 25-ton mill consists of two amalgamators working 8 hr. each. On the day shift the mill is operated by a mill foreman who takes care of cleaning the amalgam, retorting and melting.

The ore averaged \$11, the concentrator heads or copper plate tailings \$2.80, the concentrator tailings \$1.80 and the concentrates \$40 per ton.

Golden Chariot Amalgamation.—In this Idaho plant, the ball-mill pulp first flows through a rifled launder, then to a classifier the overflow of which goes over corduroy tables, followed by copper plates. In this section of the plant 84 per cent of the total gold and 11 per cent of the total silver are recovered. Flotation follows. The corduroy is washed once a day. Three copper plates have a total area of 144 sq. ft. for 100 tons of ore milled daily and are set at $1\frac{1}{2}$ in. per ft. The plates are dressed with sodium hydroxide and mercury every 6 hr.; amalgam is removed every 12 hr. by a piece of rubber belting.

Homestake Amalgamation.—Homestake experience favors amalgamation as a cheap method of gold recovery preceding cyanidation. Laboratory tests at mill sizes indicate that upward of 75 per cent of the gold is free. Of this, about 60 per cent is caught by mercury which is fed to the rod mills and ball mills in closed circuit with Clark-Todd amalgamators and Dorr classifiers in the South plant, the only one now operated.

The amalgamator is shown in Fig. 68. It provides new means of increasing the catch of amalgam, by causing the pulp to change direction several times, each change resulting in a retarded velocity of flow and affording opportunity for amalgam to build up on the plate. In this apparatus the launders are fitted with amalgamated trays, which facilitate the attachment of amalgam

particles. The trays are deep enough to retain a considerable body of amalgam without loss.

In the amalgamator the amalgam is not caught in a thin film or layer, spread over a large surface of plate; rather, the catch is three dimensional. The amalgam builds upon itself to form bodies of appreciable thickness.

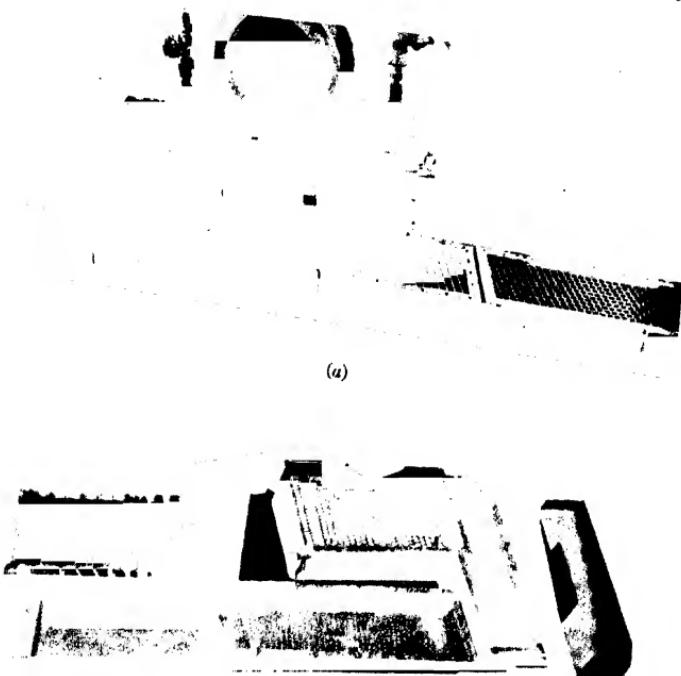


FIG. 68.—The Clark-Todd amalgamator as at the Homestake, S. D.; (a) as set up at discharge of rod mill; (b) disassembled for cleanup.

The amalgamator also removes scrap iron, fragments of the grinding media and other material at the point of discharge of the mill, thus keeping clean the recovered amalgam. It does all this in much less floor space than did other systems. The chip screen is removable, but the guard screen below it is not.

The second launder is lined with a removable amalgamated-copper box. The bottom of the third launder is covered with an

amalgamated silver-plated copper plate. This plate is also removable and is taken out and cleaned at more frequent intervals than is desirable for the trays.

Homestake amalgam yields 43 per cent gold; mercury consumption is $\frac{1}{8}$ oz. troy per ton of ore crushed, the cost of which is 1.25 of the 1.4 cts. total cost for amalgamation.

Kolar Practice.—Formerly, in India, copper plates in front of the stamp mortar boxes and in the tube-mill circuit, also

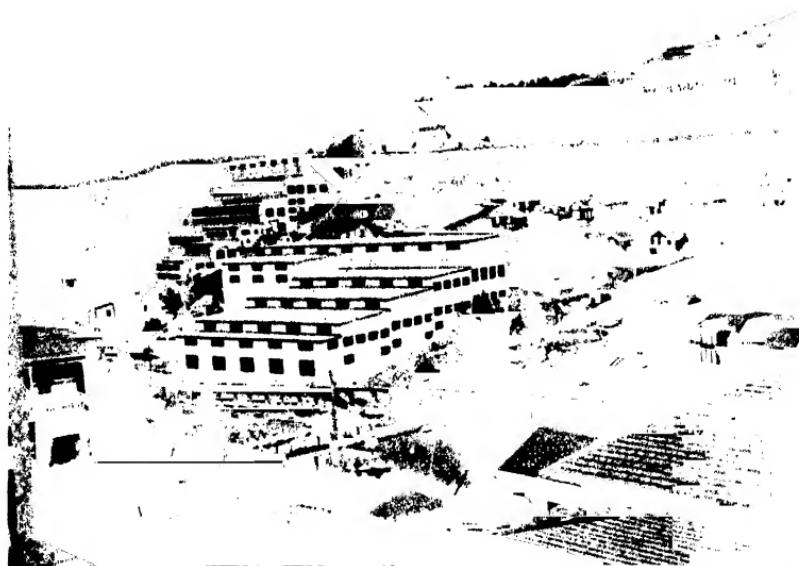


FIG. 60.—New cyanide plant 3, Homestake Mining Company, Lead, S. D. This mine has operated continuously since 1887.

mercury wells with baffle boards below the plates, saved 78 to 89 per cent of the gold. Now, as at the Ooregum mine, blanket tables below the mortar boxes and blanket machines following the tube mills, with copper-plate amalgamation of the concentrates, recover approximately as much as with all-amalgamation. In the blanket machines the blanket or carpet is attached to jute cloth or to discarded filter cloth and is washed by spraying every $1\frac{1}{2}$ to 2 hr.

Rand Amalgamation.—As mentioned under "Corduroy," amalgamation on the Rand is an art of the past, except as it concerns the use of mercury in treating corduroy and Johnson

concentrator material. Most of the plants use corduroy, and a few the Johnson machine.

CORDUROY AS A GOLD SAVER

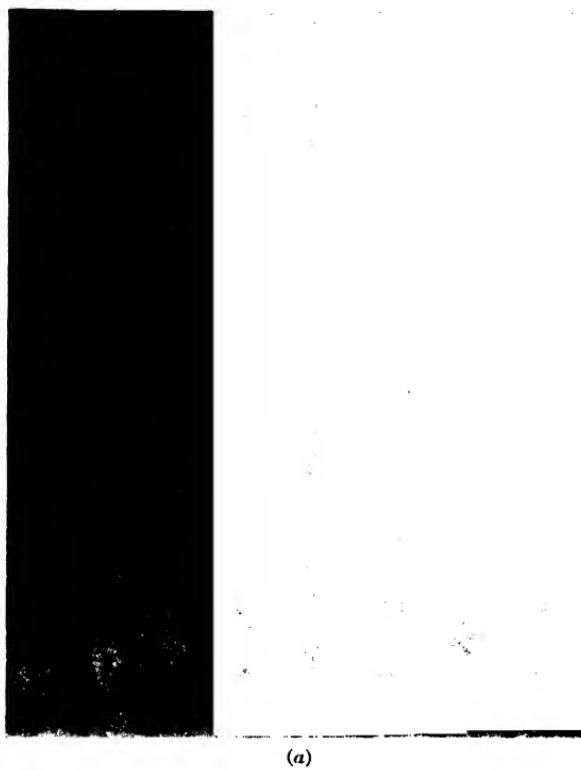
The use of blankets, canvas, coco matting and corduroy—in fact, any material with nap—to save gold not held by copper plates, also to catch some heavy materials, has persisted from the time of Agricola to the present. From this practice has come the present use of corduroy alone to entrap the gold. It has replaced amalgamation on the Rand, is satisfactorily used in Australia and in Canada and is slowly being introduced into the United States. Blankets are preferred at Kolar, India. The information on corduroy following is from the article by M. W. von Bernewitz in *E. and M.J.*, February, 1935.

Corduroy is a thick and durable cotton stuff or cloth, corded or ribbed—a pile or nap on a base. Corduroy for wearing apparel differs from that used in gold recovery in having narrower and closer cords or ribs. Also, in industrial corduroy, the nap has been cut as in the pile on certain high-grade floor rugs, thereby offering innumerable spaces for entrapping gold.

The material is known as a “pulp-sifting corduroy cloth.” It is woven from special material, and after being put through a serrating machine it has a deep pile or nap with strong backing. Then the cloth is shrunk. It is made up into bales of about 80 yd. in two widths—28 and 36 in. Figures 70 and 71 show the weave and nap of corduroy.

When to Use Corduroy.—No decided answer can be given as to the preference of corduroy or copper plates, because both give satisfactory recovery on ores containing free gold. But when such factors as initial cost, skill required to operate and maintain, the danger from mercurial poisoning, the possibility of theft of amalgam, loss of time, effect of bad water and scouring by coarse pulp and necessary general supervision are considered, corduroy seems to have the advantage. Corduroy is not expensive; it does not require skilled labor; theft is not so simple and a remarkably large flow of pulp can be passed over it. Even if an ore is ground fine, it may pay to use corduroy in the flow-sheet and catch the larger particles of gold which might prolong cyanidation.

What is known as "rusty" gold is amalgamable with difficulty, if at all. The metal may be coated with iron, occasionally with



(a)



(b)

FIG. 70.—Weave and nap of corduroy, natural size. (a) Plan; (b) end view.
manganese or other elements. Copper plates will not catch this sort of gold, but corduroy will.

Method of Applying Corduroy.—In laying corduroy, the cords or ribs are placed across the table and flow of pulp, with the high side of the nap facing the stream. Each length overlaps the succeeding length a few inches.

In general, the slope of the corduroy table must be determined by actual trial—the size and specific gravity of the solids and the liquid-solid ratio being the principal factors that influence the

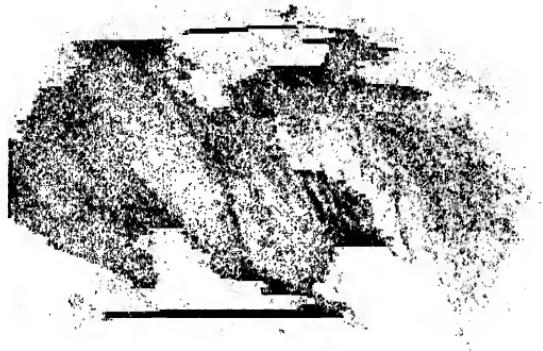


FIG. 71.—Corduroy nap, enlarged ten times.

slope. An unclassified pulp with up to 20-mesh sand at 4 or 5 to 1 dilution will flow down $1\frac{3}{4}$ in. per ft.; more dilute and finer pulp needs less slope. Roughly, 3 tons of ore passing over the tables should have 1 sq. ft. of corduroy.

Cloths are rinsed as often as may be necessary to keep the riffles from packing with heavy minerals. They are kept in use until the pile or backing gives way.

AT LAKE VIEW AND STAR

Corduroy strakes have been used for three years in the ball-milling and fine-grinding sections of the plant, as shown in Fig. 127. As no free gold can be observed in the flotation feed under

the microscope, almost complete recovery on the tables is assumed.

Raw Ore.—A sieve test of the pulp flowing over primary corduroy is 5 per cent on $\frac{1}{4}$ -in., 10 per cent on 10 mesh, 25 per cent on 20 mesh, 35 per cent on 60 mesh, 10 per cent on 100 mesh, 1 per cent on 150 mesh and 14 per cent through 150 mesh. Of the pulp passing the corduroy tables, 20 per cent is plus 200 mesh. The gold varies from microscopic grains to flat pieces up to $\frac{3}{16}$ in. in diameter.

The slope of the primary tables is 1 in 8, and of the secondary tables 1 in 10. Both sets are 3 ft. wide and are 6 and 10 ft. long, respectively. The pulp flow is 8 tons per sq. ft. per day over the primary corduroy and 15 tons over the secondary corduroy.

The wide-ribbed corduroy, costing $3\frac{1}{2}$ shillings (84 cts.) per yd., is favored. It is laid so that the deeper projecting part faces the stream of pulp. To have a series of extra-deep ribs, cloths are cut and resewn with the overlap against the stream.

All cloths are washed each morning, but the primary are changed every 12 hr., and the secondary every 24 hr. The respective lives are 6 months and 9 months.

Roasted Concentrates.—Lake View and Star discards everything except the flotation concentrate, which is roasted and cyanided. Corduroy was used to catch the gold in the pulped calcine, but it has been replaced by cut onion sacks. The calcium sulphate formed in roasting clogged the corduroy and rendered it difficult to wash. This burlap costs 6 pence (12 cts.) each sack and is of about 6 mesh.

The 3- by 10-ft. calcine tables have a fall of 1 in 10, and the pulp is 1.3 tons per sq. ft. per day of material, 100 per cent passing 200 mesh. The sacking is washed each morning and is changed every 12 hr. It has a life of 4 weeks. According to J. F. Thorn, general manager for the company, corduroy saves 23 per cent of the total gold in Lake View and Star ore.

CANADIAN PRACTICE

At Dome Mine.—The following excerpt describing the Dome blanket practice is from the paper by P. D. P. Hamilton in *Trans. 112, A.I.M.E.*

In the Dome mill, destroyed by fire in 1929, free gold was removed from the circuit by a combination of amalgam plates and blankets. In

the new mill an attempt was made to use an all-cyanide process, but this was quickly dropped, and a blanket plant installed to remove the free gold from the circuit.

The feed to the blanket tables is the unclassified product (about 60 per cent minus 200 mesh) of the primary and secondary grinding circuits. The pulp is about 5 parts water to 1 part ore. The primary load is 1500 tons, with an additional circulating load of approximately 2000 tons, making a total load over the blankets of 3500 tons daily.

There are 28 blanket tables of a combined area of 756 sq. ft., each table being 4 ft. 6 in. wide by 6 ft. long. The tables are of wood, set so that the inclination of the blankets is $1\frac{3}{4}$ in. per ft. The corduroy blankets are 28 in. wide and cut 5 ft. long to allow for shrinkage during use. The first strip is held in place by a flat iron bar, but the other two strips are held in place by lapping the upper one over the next lower one. The "special heavy-backed undyed corduroy 28 in. wide" is specially manufactured for this purpose by James Johnston, 18 London Road, Manchester, England.

The strips are placed on the table with the high sides of the cords toward the flow of the stream, thus making the necessary riffles to catch the gold. The blankets are removed and washed five times during an 8-hr. shift, three men changing, washing and looking after distribution on tables, etc. When the blankets are changed, they are folded and rolled up so as to keep the product on the inside. They are replaced by another set of blankets but before the next change are washed and rolled ready to be reused. The washing is done in boxes, one for each shift, the blankets being merely unrolled and moved longitudinally up and down in water two or three times. This removes the greater part of the product. When blankets are discarded, they are more carefully cleaned and are then treated by cyanide in a small tank until no further gold comes into solution.

The pulp is fed from launders to each table by means of a flexible-rubber pipe. When blankets are to be changed, this hose can be changed to a second launder which, by means of a small air lift, returns the pulp to the original feed stream.

Each wash tub has a pipe leading out of the bottom, so that it can be loaded directly into an amalgam barrel set directly below the tub. The amount collected in each tube in an 8-hr. shift is approximately one ton, which is the capacity of the amalgam barrels. The barrels are 31 in. in diameter by 5 ft. long and are lined with white-iron liners, but barrels with cast-iron shells are preferable because they can be used without linings. Linings invariably collect and hold up amalgam. Each barrel is charged with 500 lb. of $1\frac{1}{2}$ -in. balls and is driven at 19 r.p.m.

These barrels are loaded each day about noon; lime is added and the barrels are closed and started. The real function of the grinding is to

brighten all the gold so that it will amalgamate readily. At 6:30 the following morning the barrels are stopped, and the requisite amount of mercury is added. The barrels are then rotated again for $1\frac{1}{2}$ hr. and then dumped. The mercury and amalgam are collected, cleaned up and pressed into cakes which contain approximately 50 per cent mercury, carrying about 8 dwt. in gold per oz. The amalgam is taken to the refinery, where it is held until sufficient accumulates for a retort.

In addition to the three men on each shift, two men on the day shift handle the dumping of the barrels and the cleaning of the amalgam.

During the first six months of 1934, the total recovery at the Dome mill was 98.6 per cent, the blanket plant giving 76.1 per cent and the cyanide plant 22.5 per cent. The tailings averaged 0.1068 dwt.

At Siscoe.—Ore from the Siscoe mine, Quebec, is mostly quartz with never more than 1 per cent of sulphides. The gold is coarse and, for the most part, free. The plant treats 330 tons per day. The coarse ball-mill discharge is classified in hydraulic cones, the underflow being a concentrate the gold content of which is caught on Gibson tray amalgamators and on a silver-coated copper plate.

The overflow of the cones goes to Dorr classifiers which are in closed circuit with the mills. The classifier overflow passes to other cones, the overflow of which, containing 38 per cent solids, passes over three sets of four parallel corduroy tables, each 4 by 12 ft. with a slope of $\frac{1}{38}$ in. per ft. One set of cloths is washed every hour. The corduroy concentrates also are amalgamated and account for 10 per cent of the total of 92 per cent of the gold saved by amalgamation. In other words, the gold in cone concentrates and in corduroy concentrates is finally recovered by mercury. Cyanidation extracts about $5\frac{1}{2}$ per cent of the gold, so the total extraction is $97\frac{1}{2}$ per cent.

The consumption of corduroy at Siscoe is approximately 0.002 yd. per ton of ore, and the 36-in. material costs \$1.19 per yd. at the mine. According to O. Mathews, mill superintendent, corduroy has been found to be effective in recovering not only free gold but also the coarse sulphides. These are ground in cyanide after first being treated with mercury.

RAND PRACTICE

Review.—The revival of corduroy originated on the Rand, where copper plates had saved many millions of ounces of gold. But, according to T. K. Prentice in *Jour. C.M. and M. S.S.A.*,

September, 1933, "not all metallurgists on the Rand favor corduroy."

Except the newer plants, such as Daggafontein and East Geduld, which are all-sliming, the majority of the mills use corduroy or the Johnson concentrator. The total amount of ore treated by corduroy is $2\frac{1}{2}$ million tons per month, and the gold caught ranges from 40 to 55 per cent of the total.

The Johnson concentrator, its action and its place in a plant were described by E. H. Johnson in *Jour. C.M. and M. S.S.A.*, April, 1927. Briefly, it is a rotating drum with a rubber lining

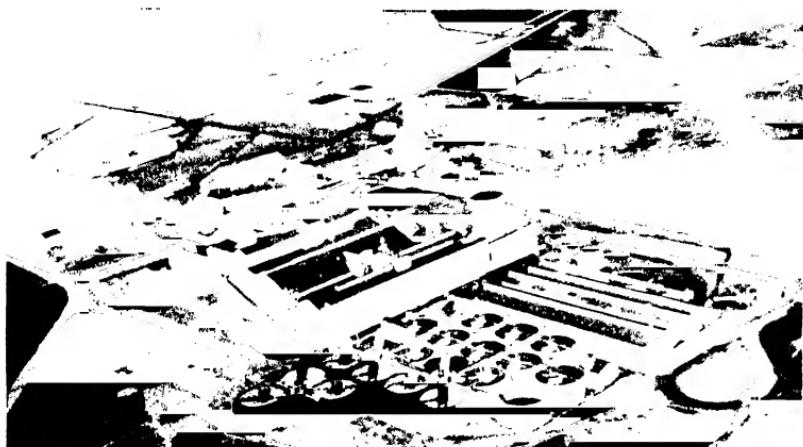


FIG. 72.—Aerial view of East Geduld Mines, Ltd., illustrating characteristic South African tailings piles and out-of-door installations of thickeners and agitators.

in which riffles or grooves are arranged as in the rifling of a gun. The sulphides collect in the riffles and as they are carried upward are removed by sprays. This machine recovers 44 per cent of the pyrite and 55 per cent of the gold. Its free gold content is removed on blanketing, and the pyrite is ground and cyanided with the mill pulp. In an East Rand proprietary plant, corduroy tables followed the Johnson machine.

Although written 12 years ago, Wartenweiler's paper in the *Jour. C.M. and M. S.S.A.*, February, 1923, is one of the few and most complete references on corduroy. It is replete with figures and flow-sheets. It is entitled "Recovery of Gold by Blanket Concentration in Substitution of Plate Amalgamation." In a

group of large gold producers, plates had been saving 47 to 73 per cent of the gold.

Plate amalgamation [to quote Wartenweiler] has gradually become encumbered with a number of disabilities. . . . With the increasing practice of fine grinding, the perfection of cyanide extraction and of precipitation, the importance of high recovery by amalgamation has receded according to the degree of efficiency of the section of the reduction plant devoted to recovery of gold by cyaniding. . . .

The Apex plant, treating ore from the Modderfontein East, was among the first to discard plate amalgamation. The copper plates, from which accumulated amalgam had been removed, were converted to corduroy tables. No holding-down device is used, but air bubbles that form are "ironed out." Even distribution of the pulp for corduroy is as important as for plates.

Wartenweiler found the following after a 4-hr. run of pulp over a series of five corduroys:

Corduroy	Gold Panned and Weighed, Per Cent
1	76 to 80
2	11 to 18
3	3.1 to 4.2
4	2.3 to 3.7
5	0.8 to 2.1

Current Use of Corduroy.—The corduroy tables are generally placed immediately after the tube-mill discharge. This pulp will carry 30 to 40 per cent of plus 90-mesh material.

The average area of a corduroy table is 5 ft. wide and 10 or 12 ft. long. The slope is 2 in. per ft.

Washing of the corduroy is done every 4 hr., but the first strips may be washed twice as often. Its life varies but may average 35 days. It costs $4\frac{3}{4}$ shillings (\$1.14) per yd. of 36 in. width. Probably, 36,000 yd. is used annually on the Rand. The charge against milling is 1 or 2 pence (2 or 4 cts.) per ton.

Some metallurgists do not regard amalgamation or corduroy concentration as necessary on the Rand with adequate fine grinding. Its use, however, does reduce the lockup of gold in tube-mill circuits. At some plants, enough osmiridium is recovered as a by-product to pay for the whole process of using corduroy.

Where corduroy concentration is employed on the Rand, the discharge from the tube mills passes over an area of corduroy equivalent to approximately 1 sq. ft. of corduroy per ton of ore milled per day. The amount of concentrate is about 1 ton for every 1000 tons of ore milled. This is reconcentrated to a half and then amalgamated in a barrel. The consumption of mercury during 1933 was 0.015 oz. per ton milled, which is one-tenth of that in 1910 when plate amalgamation was in vogue, according to T. K. Prentice.

UNITED STATES PRACTICE

At Golden Cycle.—In the cyanide plant of the Golden Cycle mill, blanket tables are used after grinding.

Roasted ore, as well as iron concentrate, is ground in Chilean mills and passed over 6- by 12-ft. tables, each with two 3-in. drops. A light Canton flannel is used on the tables. Blankets are changed every 12 hr., the concentrate going to amalgamation pans. Cyanide solution is used in the pans. The tails from the pans pass over more blankets and return to the cyanide circuit while the concentrate is returned to amalgamation. Gold recovery in this step varies from 22 to 40 per cent.

At Montezuma-Apex.—J. B. Hutt in *E. and M.J.*, April, 1935, tells of the use of corduroy or pulp-sifting cloth at the Montezuma-Apex mill near Nashville and Placerville, Calif. The ore consists of quartz in which the gold is free and in iron sulphides associated with other minerals. It assays 4 to 6 dwt. per ton, and 230 tons are milled daily. Recovery is 94 to 95 per cent, 74 per cent of this being as bullion and 26 per cent as concentrate. After ball milling to 65 mesh and passing through Dorr classifiers, the pulp leaves some of its gold in hydraulic traps and rifled launders in the mill-classifier circuit. Corduroy is used in three places in the plant: Following the classifiers, 30 sq. ft.; following the conditioner tank, 240 sq. ft. and following the Booth-Thompson flotation machine, 100 sq. ft. Also 30 sq. ft. of corduroy is used in a periodic cleanup of the classifiers. The material from the traps, riffles and corduroy is collected and treated twice a month. It is run over a Wilfley table, and this concentrate is amalgamated in a 24- by 48-in. barrel. The contents are discharged on to a mechanical batea, from which a fairly clean amalgam is recovered. The residue flows

over silvered plates in a launder and through a mercury trap and is then returned to the table. The table tailings go to the mill circuit.

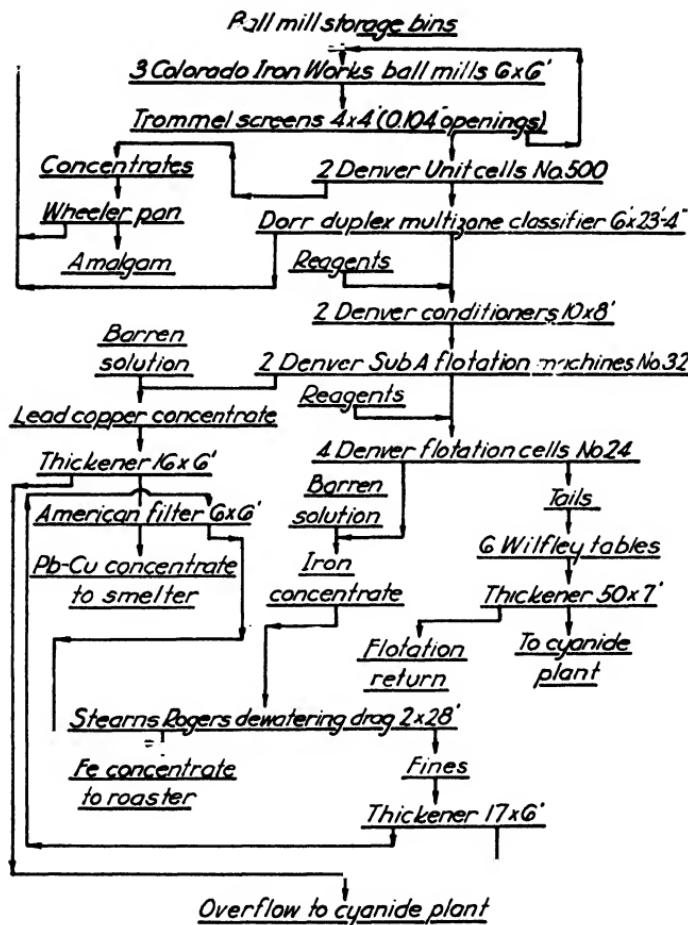


FIG. 73. Golden Cycle concentrate plant.

Magna Plant.—In January, 1933, at the Magna Plant, Utah Copper Company, according to E. W. Englemann in *M. and M.*, August, 1935, burlap or coco matting was placed in the bottom of launders handling various products from flotation. This was soon followed by placing various materials such as coco matting, cotton twill and common 10-oz. burlap on the bottom of the

main tailing launder. The burlap was found to be the most economical.

As a result a plant to treat about 12,000 tons of tailing per day was built at a cost of about \$25,000, which was considerably more than if the plant had been built on level ground.

The plant consists of two separate units each having four sets of double launders about 500 ft. long fitted with burlap.

The cleanup cycle is 40 days. Operations have not continued long enough to determine what the saving in gold will be.

When a cleanup is to be made, the burlap is washed then removed, dried and burned. The concentrate and ashes are shipped to the smelter.

TREATMENT OF CORDUROY CONCENTRATES

Barrel amalgamation is the simplest and most common method of treating the rich concentrates caught by the corduroy. A heavy, cast-iron barrel with manhole, revolving slowly, is used. In it are placed a charge of concentrates, steel balls or a rod or two, some water, lime and mercury. The whole may be run 2 to 12 hr. The pulp is discharged; the amalgam is then caught in riffles and the fine pulp in boxes or tubs, from which it may be fed slowly into the mill circuit for cyaniding. The amalgam is retorted in the ordinary manner.

The final handling of worn-out corduroy is to burn it and treat the ash separately. It should not be fed to the mill circuit for fear that the carbon may precipitate the gold from the cyanide solutions. At the Dome mill the corduroy is dumped loosely into a tank in which cyanide solution is circulated for several days. This dissolves any fine gold enmeshed in the fabric. Then the cloths are discarded.

PRECIPITATION

Five methods of precipitating gold and silver from cyanide solutions have been used: aluminum, charcoal, sodium sulphide, zinc and electrolytic. The first and third were specially developed for the silver ores of Cobalt, Ontario; charcoal has been used in Australia, with some possibility of revival; and zinc, either as dust or as shavings, has been used from the beginning of the cyanide process and continues to be the standard method used

throughout the world. In this section is given the technique of the processes and their application in certain mining centers.

PRECIPITATION BY ZINC

Zinc shavings and zinc dust are both used for precipitation of precious metals. Although most new cyanide plants adopt zinc-dust equipment, and some old plants change from zinc shavings to zinc dust, zinc shavings probably will be used at many small mines and tailings operations. Zinc dust, however, is generally more effective and satisfactory than zinc shavings and is approximately 5 cts. per ton of ore cheaper than the older method.

Chemistry of Precipitation.—Chemists differ somewhat on the theory of precipitation with zinc as to whether nascent hydrogen, liberated by the action of an alkali cyanide on zinc, has a direct effect in the precipitation or is only an auxiliary action taking place at the same time. The following facts are basic:

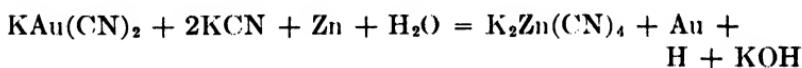
Gold and silver are electronegative to zinc in cyanide solutions and should therefore precipitate them.

Precipitation takes place only in the presence of free cyanide.

Precipitation is always accompanied by the liberation of hydrogen.

The alkalinity of the solution is increased during precipitation.

Clennell states that the entire effect of the precipitation of gold may be expressed by the equation



but that the reaction between zinc and cyanide takes place independently and bears no necessary proportion to it. The following equation represents the probable reaction:



For more details see *Cyanide Handbook*, by J. E. Clennell; *The Cyanide Process of Gold Extraction*, by James Park, and *Manual of Cyanidation* by E. M. Hamilton.

PREPARATION OF PREGNANT SOLUTIONS FOR PRECIPITATION

The requisites of effective precipitation of gold and silver from cyanide solutions are briefly as follows, based on notes supplied by L. D. Mills of the Merrill Company:

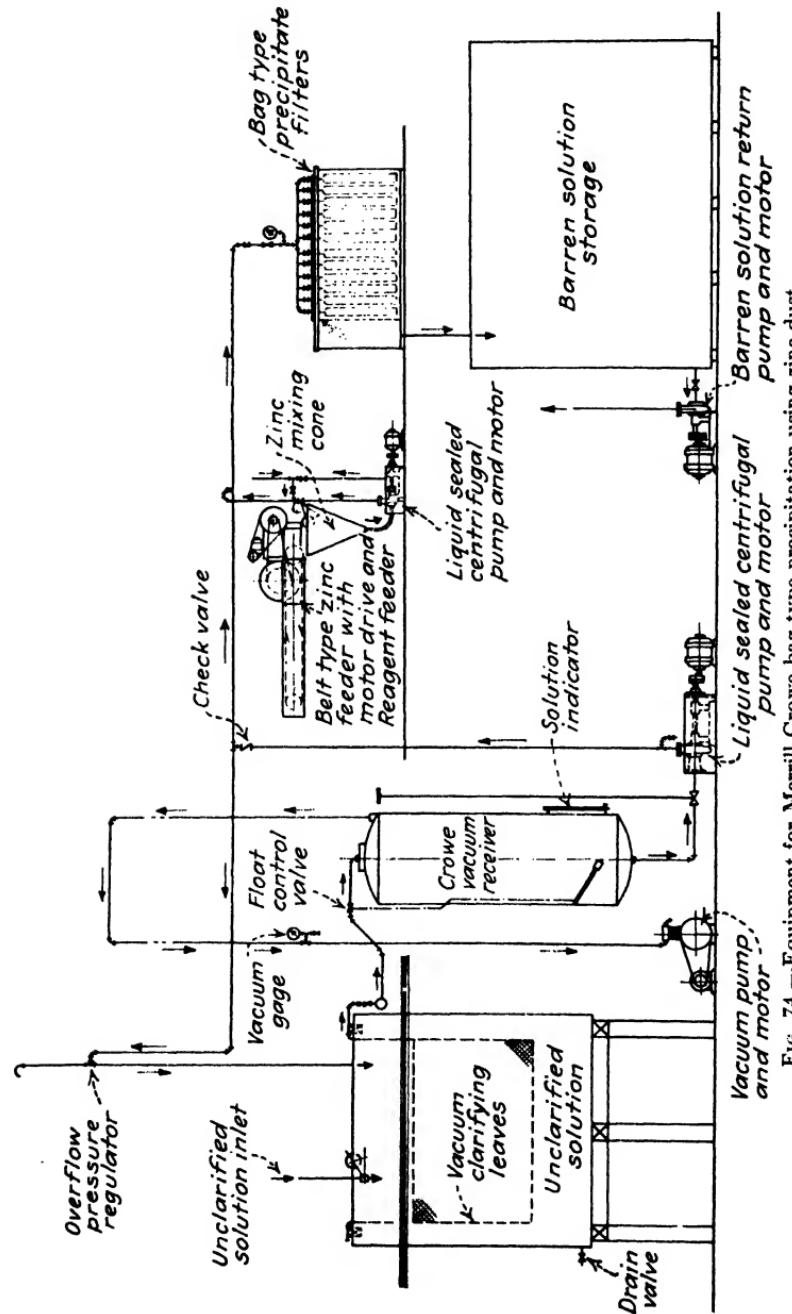


FIG. 74.—Equipment for Merrill-Crowe bag-type precipitation using zinc dust.

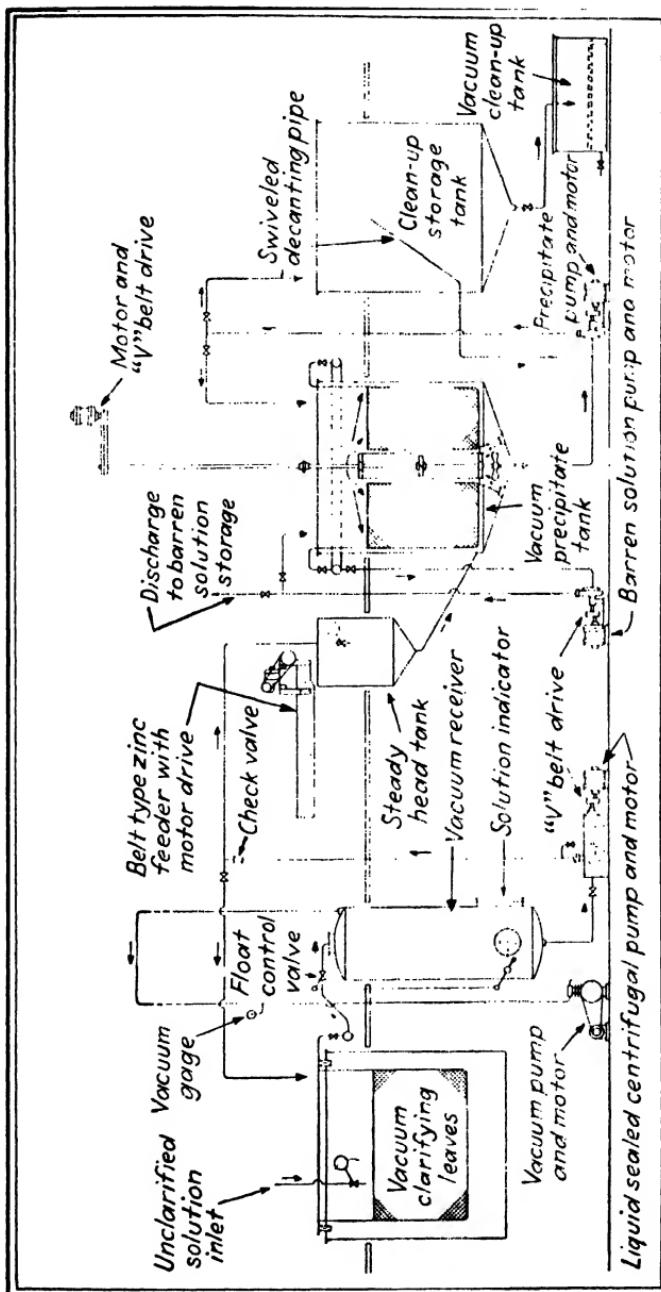


FIG. 75.—Equipment for Merrill-Crowe vacuum-leaf precipitation with zinc dust.

Clarification.—The first essential to effective precipitation is clear solution. Regardless of color, the solution must be bright and sparkling and entirely free from colloidal solids. With proper skill and attention it is possible to obtain such clarification with gravity sand filters, with plate-and-frame pressure filters or even with vacuum-leaf filters.

If canvas is the filter medium, either under pressure or under vacuum, it should be stitched around the periphery of the leaf, and the stitched area painted with P and B or a similar paint. In starting a new or recently cleaned filter unit, irrespective of type, the effluent should be returned, for a few minutes at least, to the unclarified storage.

Close attention to these points will insure a lower tail solution, less consumption of precipitant, and much more satisfactory melting and refining.

Deaeration or Removal of Oxygen.—Efficient and complete precipitation of metals from cyanide solutions requires the preliminary removal of dissolved oxygen. The efficacy of the zinc-dust process is due largely to the preliminary removal of dissolved oxygen from the solution and subsequent prevention of reabsorption of oxygen in the solution.

Gold-bearing solutions on the Rand are clarified by passing them through sand clarifiers having at least 2 sq. ft. of area per ton of solution in 24 hr. Although the Crowe vacuum process is the most efficient for removal of oxygen, reducing the oxygen from 6.5 to 0.5 milligram per liter with a vacuum of 22 in., the chemical method of oxygen removal is still used at some mines. To the sand clarifiers mentioned are added fine iron and highly pyritic sand. The solution is deprived of much of its oxygen as it percolates through the bed of sand, especially if two clarifiers be placed in series.

Chemical Control.—For effective precipitation, solutions must contain enough free cyanide to dissolve the requisite amount of zinc and to hold in solution the compounds that are formed when zinc dissolves in alkaline cyanide solutions. This result is frequently best obtained by adding a drip of strong cyanide solution to the zinc emulsion zone, when zinc dust is used.

Efficient precipitation of gold and silver solutions by means of the Merrill-Crowe process is generally independent of the strength of the solutions in cyanide and alkali. Substantially

complete precipitation is obtained in some plants where solutions contain no more than 0.05 lb. of either NaCN or CaO per ton of solution. In cyaniding silver ores, solutions frequently contain as high as 5 lb. NaCN per ton, with protective and total alkali equivalent to several pounds CaO per ton solution.

In cyaniding most gold ores, the lime consumption is generally dictated by the requirements of effective settling and is almost invariably higher than necessary for the best precipitation. In cyaniding silver ores, a high alkalinity is needed to dissolve the minerals, and lime consumption for this purpose coincides with that needed for settling.

The principal detrimental effect of high cyanide and alkali is to consume zinc wastefully. An excess of lime in the solutions will sometimes coat the zinc and choke the filters, rendering frequent cleanups necessary.

Precipitation of all gold solutions and of some silver solutions is facilitated by the addition of a soluble lead salt to the solution. Either lead nitrate or lead acetate may be used, although the former is preferable. The amount of the lead salt approximates 10 per cent of the weight of zinc dust if this is added to the solution. The dissolved lead salt is added in the form of a continuous drip to the zinc-emulsion cone or mixing tank or may, under certain conditions, be added to the solution entering the clarifying tank but never with the zinc dust. The lead precipitates as a thin metallic film on the zinc, thus creating an active galvanic couple, with usually more rapid and complete precipitation of the gold and a lower zinc consumption. For some silver solutions the lead salt should always be added before clarification, because in most solutions some of the lead is immediately precipitated as an insoluble basic salt which rapidly clogs the precipitation filters.

The successful use of lead salts requires careful supervision because the addition of an excess at any time may coat the zinc with enough lead to retard, or even prevent, galvanic action. This explains why lead salts are not used in precipitating solutions containing considerable amounts of silver, copper or lead, there being sufficient silver or base metal present to form an effective couple with the zinc.

As a precipitant activator, the Merrill Company has determined that sodium bisulphite is of practical benefit and is testing

it on a plant scale. Excess alkalinity must be neutralized to about pH 6.6. One-tenth pound sodium bisulphide is required for a ton of cyanide solution.

ZINC-DUST PRECIPITATION

For many years precipitation by zinc dust, as by the Merrill-Crowe process, has been recognized as the most efficient and economical method of precipitating gold and silver from cyanide solutions. Embodying precipitate filters of the plate-and-frame type, of the vacuum-leaf type or of the more recently developed pressure bag-filter type, the process is in use in the great majority of cyanide plants throughout the world (see Figs. 74 and 75).

Advantages.—As compared with zinc shavings, more uniform and efficient precipitation is obtained because a fresh surface of precipitant is being constantly exposed to the solution. Where a base metal such as copper or alkaline earth-metal derivatives such as lime, magnesia or alumina are present in the solution in large quantities, zinc shavings become quickly insulated or polarized, and the boxes must be frequently cleaned, and fresh zinc added.

Calcium sulphate in particular is always present in the treatment of sulphide ores and rapidly coats zinc shavings, rendering them inert. In the zinc-dust process the time of contact is so short and the flow of solution through the zinc so rapid that this coating is reduced to a minimum. Even though small, unconsumed particles of precipitant may become coated with copper or sulphates, the amount of zinc thus rendered inert is negligible and is constantly replaced by the addition of fresh, active precipitant.

A difficulty frequently encountered in the precipitation of solutions containing dissolved oxygen, particularly in operating zinc boxes in relatively cold climates, is the formation of hydrated zinc oxide or so-called white precipitate. This coats and rapidly destroys zinc and, being mixed with calcium sulphate, is insoluble in acid and causes endless trouble in refining. This compound cannot form in the absence of free oxygen and is therefore entirely absent in plants using zinc dust after deaeration.

Another reason why such uniform and efficient precipitation is obtained by zinc dust is that the filter cloths are at all times coated with a layer of fine precipitant and precipitate, and no

particle of solution can pass through the filter without first coming into intimate contact, in fact almost molecular contact, with the precipitating agent.

It is this extremely fine state of subdivision of the precipitant that renders zinc dust so efficient. For a given weight of metal the effective surface of zinc dust exposed is many hundred times that of shavings, and with this large area a very brief contact between the zinc and the solution is sufficient for complete precipitation of the metals.

Chemical Considerations.--To obtain perfect precipitation, each molecule of metal-bearing solution must be brought into contact with a particle of precipitant, must give up its metal and immediately thereafter must be removed from contact with other metal-bearing molecules. This condition cannot be met in a zinc box, and the resultant diffusion accounts for the long boxes necessary and the usual incomplete precipitation. In the zinc-dust process, however, this requirement is fulfilled perfectly, as the solution passes through the layer of finely divided precipitant deposited on the surface of the filter cloth.

Under the right conditions, nascent hydrogen is freely evolved throughout the layer of the precipitant, thus creating the reducing condition necessary for the precipitation of the metals. The actual deposition is due to the action of galvanic couples, consisting of hydrogen zinc, gold zinc, silver zinc, lead zinc and sometimes copper zinc. The precipitation is closely analogous to electrolytic deposition, with the exception that in the zinc-dust process it is possible to obtain trace barrens because diffusion or mixing of the impoverished solution with the unprecipitated solution is prevented. Polarization of the cathode particles is minimized by the rapid flow of solution which carries the molecular hydrogen along with it.

The precipitation of gold from cyanide solutions with zinc requires either enough contact or enough caustic alkali or both to attack the metal with the evolution of hydrogen. Also, the zinc must be in such a form that each tiny bubble of nascent hydrogen will make contact with and adhere to a particle of zinc, forming an active couple. Obviously, this requirement is met much better within a layer of zinc powder than upon the coarse filaments of metal in a zinc box. Therefore, a much higher efficiency of the evolved hydrogen is obtained in the Merrill-

Crowe process than with zinc shavings. If the solutions contain dissolved oxygen, the first hydrogen generated is wasted in combining with this oxygen; this, of course, involves a corresponding loss in zinc and alkali and is entirely obviated in the zinc-dust precipitation process, which removes all dissolved oxygen from the solution before contacting with the precipitant.

Economics.—The foregoing means that less zinc is dissolved per unit of gold precipitated or deposited. Less unconsumed zinc is left (10 to 15 per cent) in the precipitate; therefore melting and refining charges are less. In silver precipitation the unconsumed zinc is less, being only 3 to 5 per cent. Only 0.6 oz. Merillite or zinc dust per ounce silver is used, compared with 2 oz. when zinc shavings are used.

Most of the zinc dissolved in cyanide solutions ultimately goes to form a zinc cyanide, and each pound of zinc combines with 3 lb. sodium cyanide. Subsequently, when this solution comes in contact with fresh lime added to the ore during treatment, part of this combined cyanogen is regenerated—probably less than half, but at least 1 lb. of cyanide for each pound of zinc dissolved. Hence, any method that reduces the zinc dissolved in the solutions must also be responsible for a material saving in cyanide consumption. The cleaner (less foul) solutions should also result in a higher extraction of metals from an ore.

Equipment.—When first introduced, the zinc-dust process utilized the Merrill sluicing-clarifying filter for pregnant or gold-bearing solutions, the Crowe vacuum tank, a zinc-dust feeder and the Merrill triangle-shaped plate-and-frame pressure filters in which to collect the zinc-gold-silver precipitate. Many installations of this well-known equipment are in use (see Figs. 74 and 75).

Late in 1932, the Merrill Company announced a new form of equipment, the simultaneous clarification-precipitation type. This new type is now employed in capacities ranging from 100 to 1500 tons of solution daily.

Clarification and deaeration of the solution are followed by the immediate addition of zinc and precipitation of the metals without rest and without exposing the solution to atmospheric contact. Most cyanide solutions, after clarification, will, upon standing even a short time, throw out suspended colloids, consisting largely of the hydrates of alumina, magnesia and iron.

Although hardly visible to the naked eye, enough of these precipitates frequently form to coat and "insulate" the zinc, increasing the pressure in the filters and seriously interfering with precipitation. This difficulty is minimized and in most cases entirely prevented by simultaneous clarification, deaeration, and precipitation, which costs 1 to 2 cts. per ton of ore treated.

In either the bag or the leaf type of filter, a single, liquid-sealed centrifugal pump effects the successive steps of clarification and deaeration. The clarifying tank is kept filled to a constant level with unclarified gold-bearing solution, the inflow to the tank being controlled by an automatic float valve. Suspended in this tank are the vacuum clarifying leaves, with outlets connected to a manifold, which in turn is connected to the top of the vertical vacuum tower in which the solution is deaerated. Inflow of solution to the tower, and, therefore, the solution level within the tower, is controlled by an automatic float valve. Within the tower the solution passes down over suitable grids, which break up the flow into small streams and films, thus effecting the substantially complete removal of dissolved oxygen. The top of the deaerating tower is connected with a dry vacuum pump which maintains a high vacuum within the tower and removes the air released from the solution. The clarified, deaerated solution is withdrawn from the bottom of the deaerating tower by a single-stage centrifugal pump, submerged in solution to prevent reentry of air through the pump gland.

Where bag precipitate filters are used, as shown in Fig. 74, zinc dust is introduced as the solution flows from the pump to the filters. A belt-type zinc feeder, with motor drive, discharges a regulated amount of zinc dust into a mixing cone. A liquid reagent feeder, operated by the same motor drive, supplies the corresponding and uniform feed of lead nitrate solution to the cone, which is connected to the solution supply tank. The lead-zinc emulsion is withdrawn from the cone and forced into the main solution line by means of a small motor-driven, liquid-sealed centrifugal pump.

The clarified, deaerated solution now containing the proper amount of precipitant is forced through the submerged bag filters, the precipitate of the metals remaining within the bags, and the barren solution flowing over a measuring weir into a storage tank whence it is pumped for re-use.

In the discharge line of the pump which withdraws solution from the deaerating tower is an open by-pass permitting return of solution to the clarifying tank. This by-pass line is carried up to a height which automatically limits the maximum pressure on the bag filters and also provides a means for recirculating solution to the clarifier, while precoating and starting new or washed clarifier leaves. This by-pass, being open to the atmosphere, also provides means for automatically releasing the vacuum which might otherwise collapse the filter bags and cause a siphoning back of precipitate, in the event of sudden power failure.

Cleanup is effected by emptying the precipitation tank, draining and then disconnecting the bags and removing the inner filters containing the precipitate. The precipitate is dried, fluxed and melted in the usual way. The inner bags may be burned and added to the precipitate or washed and re-used.

In the vacuum-leaf precipitate filters, the zinc dust and lead solution are similarly added to a mixing agitator, which overflows into the steady head tank supplying the vacuum filters. The mixture of solution and precipitant is continuously circulated over the filter leaves, the barren solution being drawn through the filter leaves by a centrifugal pump which in turn discharges to a suitable barren-solution storage tank.

ZINC-SHAVING PRECIPITATION

As precipitation on zinc shavings is so well-known and understood, no extended reference is needed here. One feature of zinc-box precipitation is that the whole operation can be seen at a glance—solution flow, effect of lead salts, whether copper is coming down, formation of zinc white and generation of hydrogen. Precipitation on shavings is efficient and reasonably low in cost, but it offers chemical and manual problems not arising in precipitation on zinc dust.

Copper and all of its compounds readily dissolve in cyanide solutions and form an adherent deposit on the zinc. The unsuspected presence of copper in an ore is revealed by zinc shavings' becoming red.

Precipitation on zinc shavings is sometimes hindered or prevented by the formation of so-called "zinc white," the cyanide

and hydrate of zinc. It is insoluble in water but soluble in cyanide solutions and acids.

PRECIPITATION AT KOLAR

At Kolar, India, a large group of mines is still using zinc shavings for precipitation. The shavings are cut locally from 20-gage sheets. All solutions are clarified. One plant, for example, has three zinc boxes with six compartments each, five of which are used. Each compartment has a capacity of $12\frac{1}{2}$ cu. ft., or $187\frac{1}{2}$ cu. ft. in 15 cells. A total of 620 tons is precipitated in 24 hr. Average solution feed in one mil assays 41.4 gr. gold per ton. Of this, 27.5 per cent is precipitated in the first cell, 60.4 per cent in the second, 8.7 per cent in the third, 2.2 per cent in the fourth and 0.7 per cent (0.2 gr.) in the fifth - 99.5 per cent in all. A drip of a 2 per cent solution of lead acetate is added to the boxes. Zinc consumption is 0.089 lb. per ton ore treated. Most of the zinc boxes are built of concrete. Box compartments are connected by branch pipes to a main sublevel central pipe connection to a vacuum-filter tank. Wooden plugs (Fig. 76) in each cell control the flow through the branch pipes to the vacuum filter when cleaning up. The cleanup proceeds along standard lines and is done every tenth day. Between cleanups the zinc boxes are rarely dressed. Precipitate is roasted in an oil-fired muffle, fluxed with niter, manganese dioxide, sand, borax glass and sodium carbonate and melted. The bullion is 985 fine. The foregoing is from R. H. Kendall and A. F. Hosking's paper of 1924 and is substantially present practice.

PRECIPITATION ON THE RAND

Zinc shavings and zinc dust are both used to precipitate gold on the Rand, all new plants using the latter. Deaeration of

COMPARISON OF PRECIPITATION METHODS ON RAND

Condition	Zinc shavings	Zinc dust
Assay of precipitated solution, dwt. per ton	0.015 to 0.02	0.015 to 0.020
Cyanide strength to precipitation, per cent	0.011 to 0.028	0.014 to 0.016
Alkalinity (lime), per cent	0.005	0.018 to 0.02
Zinc consumption, lb. per ton milled	0.14	0.05 to 0.06
Solution precipitated per ton treated	1.3 to 2.0	1.5 to 1.8

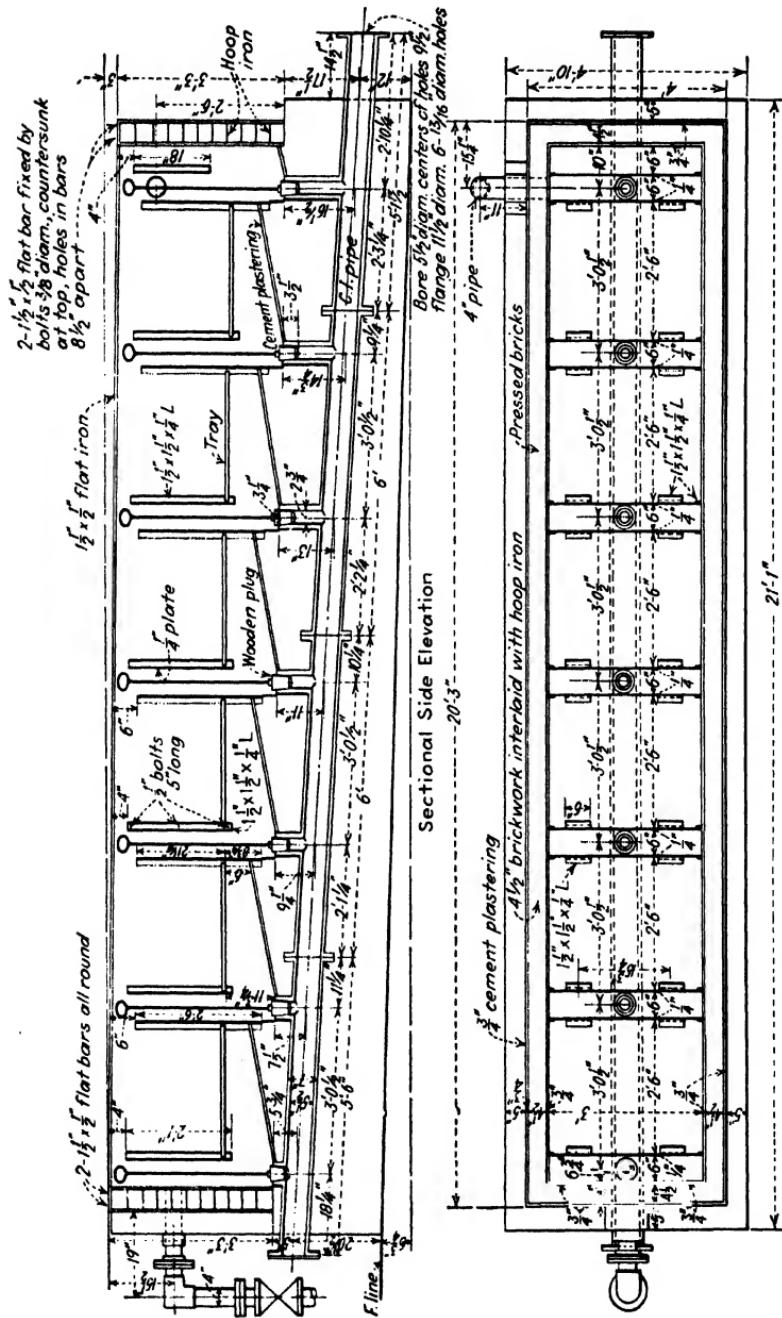


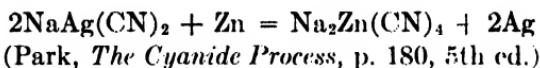
Fig. 76.—Extractor box, Mysore Mine, India.

pregnant solution for zinc shavings has eliminated the white precipitate of hydrated zinc oxide. Lead salts are added before precipitation. Wartenweiler, in *Trans. 112, A.I.M.E.*, 1934, summarizes the two methods as follows: During 1933, 1,055,-000 lb. zinc dust and 3,129,000 lb. zinc shavings were consumed.

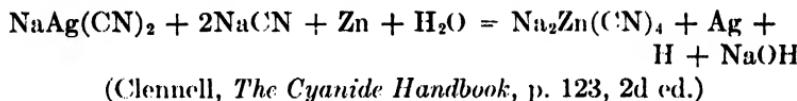
PRECIPITATION BY ALUMINUM DUST

Precipitation of the precious metals from cyanide solution by aluminum differs from the precipitation by zinc in that aluminum does not replace the precious metals in the cyanogen compound.

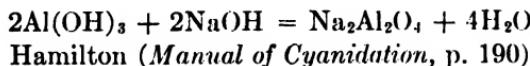
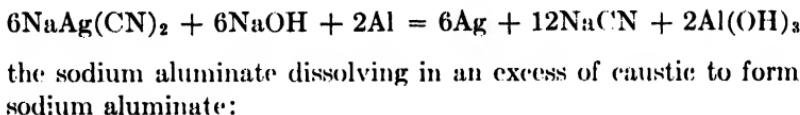
In the case of zinc the reaction may be expressed by the equation



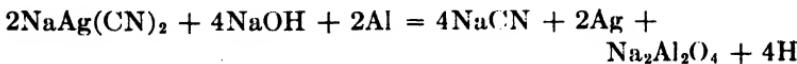
or



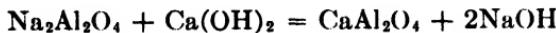
When aluminum is used, Moldenhauer, who patented this method in 1893, suggested the following equation:



suggests that the following may represent more nearly the actual reaction based upon plant observation:



It is seen that the presence of caustic soda is essential when aluminum is used. Furthermore, as a matter of practical operation lime must be absent at the time of precipitation; otherwise the following reaction will take place:

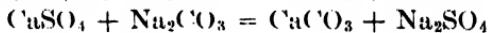
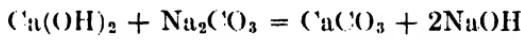


The calcium aluminate so formed would contaminate the silver precipitate and result in a low-grade product extremely difficult to flux and melt into bullion.

After precipitation, however, when the barren solution is re-used in the grinding and agitation circuits in the presence of lime, the aluminum is precipitated as calcium aluminate and removed from the plant with the tailing, caustic soda being formed.

At the Nipissing mill where only a small amount of lime was required and where the solution was already high in caustic from the preliminary desulphurizing process, no trouble was experienced with the formation of calcium aluminate in the press.

In order to overcome the ill effects of lime in aluminum precipitation when treating ores where the use of a fairly high amount of lime is necessary to promote effective settling, Hamilton and Crawford devised a treatment at the Butters Divisadero mine based upon the following reactions:



(Hamilton, *Manual of Cyanidation*, p. 195)

It allows the use of all the lime necessary for neutralizing and settlement; it yields a lime-free solution, for precipitation, and incidentally manufactures the caustic soda necessary for that operation.

Apparently, aluminum dust is not effective as a precipitant for gold alone, although the gold in solutions which contains 2 oz. silver or more per ton is almost completely precipitated. It has proved its advantages in the treatment of certain silver ores containing arsenic and antimony. It has not been widely applied, however.

PRECIPITATION BY SODIUM SULPHIDE

At Cobalt, Ontario.—The practice of precipitation by sodium sulphide was developed in 1916 at the Nipissing mill to replace aluminum precipitation. The change was necessary because of the changes in economic conditions wrought by the World War. Before that period, aluminum dust cost 33.82 cts. per lb., and caustic soda 2.11 cts. per lb. at the mine. In May, 1916, the

price of aluminum dust was advanced to 90, and that of caustic soda to 5.77 cts. per lb. At these prices, the increased cost of these two chemicals amounted to about \$33,000 a year. Although developed to meet an emergency, the use of sodium sulphide has been so successful that it will be retained permanently.

The sodium-sulphide process involves the precipitation of the silver as silver sulphide, the reduction of the precipitate to metallic silver by a desulphurizing treatment and the melting down of the resultant silver to a fine bullion.

At the Nipissing, precipitation was effected in two wooden tanks, 5 by 6 ft., provided with mechanical agitation. In the first tank the clarified solution met a small stream of concentrated sodium sulphide, which threw down the silver sulphide as a fine precipitate. To avoid blinding the canvas, the precipitate is caused to agglomerate by agitating it in a second tank before it is drawn off to the filter press. In practice it is found that 0.06 lb. sodium sulphide (60 per cent strength) is required to precipitate 1 troy oz. silver. Gold is not precipitated at all, nor is copper, if the solution contains 0.15 per cent or more free cyanide. As with aluminum, precipitation by sodium sulphide regenerates all the cyanide combined with the silver in the pregnant solution. The reactions involved are shown in the equation



At the Nipissing the precipitate is reduced to metallic silver by a modified form of the Denny desulphurizing process. The precipitate of silver sulphide is transferred to a 7- by 5-ft. iron tank provided with a mechanical agitator. Aluminum ingots weighing 500 lb. are thrown in, and caustic soda is added, about 0.03 lb. 76 per cent NaOH being required for each ounce of silver. With a dilution of 4 to 1 the solution has a strength of about 8 per cent NaOH. The mixture is agitated until the black silver sulphide turns brown. This requires about 8 hr., depending upon the temperature of the solution. The caustic solution must not be too hot, as the reaction with the aluminum will then become too violent. This will interfere with the reduction of the silver sulphide, as the large amount of hydrogen given off prevents the actual contact necessary between the sulphide and the aluminum. The desulphurized precipitate is collected in a filter press, washed free of sodium sulphide and delivered to the

refinery. The aluminum ingots remaining are left in the bottom of the tank for the next charge.

PRECIPITATION ON CHARCOAL

T.P. 378, 1927, U.S.B. of M., by John Gross and J. W. Scott, is a most comprehensive publication on the use of charcoal as a precipitant. It lists 93 references to the literature since 1891 and 7 between 1830 and 1890. Although these items are scattered, doubtless charcoal has been most used in Australia where, at certain times and in certain interior places, zinc has been expensive. That condition no longer exists, for Australia now produces all the zinc shavings and dust required. Gross and Scott briefly review what had been done prior to their research in about 1926 and then detail their many careful experiments. Their findings are as follows:

1. The mechanism of the precipitation involves adsorption accompanied by a chemical change.
2. Precipitation of silver on charcoal from cyanide solution follows the same laws as precipitation of gold, although it is slower; charcoal has less capacity for silver than for gold.
3. The limit of charcoal precipitation from cyanide seems to be about 2000 oz. of gold and 1000 oz. of silver per ton of charcoal.
4. Little difference exists among charcoals prepared from different woods.
5. The most important point in the making of charcoal is the heat treatment, either during the making or subsequent thereto.
6. To quench charcoal does not improve it.
7. Pulverization finer than 200 mesh does not appreciably add to the efficacy of charcoal.
8. Few substances in the solution appreciably affect precipitation.
9. The adsorbed gold or silver salt is soluble to some degree in boiling water and is especially soluble in hot cyanide.
10. There is a possibility of so changing the adsorbed gold or silver salt on charcoal that the charcoal may be used for further precipitation.
11. Precipitation of gold on charcoal from cyanide is not metallic and has not the chemical properties of the metal. No gold is visible, even when observed under the microscope.
12. Few substances in solution have a bad effect on precipitation of gold or silver on charcoal, but sodium sulphide and free cyanide decrease the rate.
13. There is a loss of cyanide in charcoal precipitation, due to adsorption.
14. Precipitation of gold is effective from low-grade solutions, but silver is slower, and a countercurrent method is proposed.
15. Some regeneration of cyanide is possible from charcoal when sodium sulphide is used as a "fixer."

16. Charcoal could replace zinc when foul solutions cause trouble in precipitation.
17. A small, isolated plant having wood available could employ charcoal in preference to zinc, using three stages with fairly coarse charcoal.
18. The charcoal has to be burned; and to avoid loss by dusting in doing this, it could be impregnated with sodium carbonate.
19. Charcoal will precipitate gold or silver from a cyanide-ore pulp; the charcoal can then be separated from the pulp by flotation.

With regard to findings 1, 9, and 10 wherein "adsorption" and "adsorbed gold" are mentioned, Gross and Scott credit A. W. Allen for advancing the theory that adsorption without chemical change of the alkaline aurocyanide was the correct explanation. This was generally accepted, and all known facts seemed to bear him out. Allen's discussion is to be found in *Trans. I.M. and M.*, 1917-1918, in Vol. 18 of *M. and C.E.*, 1918 (now *C. and M.E.*); in Vol. 106 of *E. and M.J.*, 1918 and in Julian and Smart's *Cyaniding of Gold and Silver Ores*.

The use of charcoal as a precipitant preceding flotation is covered by Australian Patent 15635, June 21, 1934, issued to V. T. Edquist, manager at the Sons of Gwalia mine, Western Australia. The specification reads:

The invention consists broadly in dissolving the soluble gold or silver contents in cyanide or other solvent, adding an excess of a precipitant such as powdered charcoal and finally subjecting the pulp to flotation separation so as to obtain a float concentrate rich in precious metals.

The gold and silver may be recovered from the flotation concentrate containing the charcoal, and precipitated gold and silver by any well-known means.

In treating ore containing soluble gold and silver contents and also gold and silver associated with pyritic minerals, it will be obvious that unless these minerals have a destructive or inhibiting effect on a cyanide, the direct method of treatment as above (two experiments described) will be satisfactory. If, however, minerals (capable of flotation) are present, which have a destructive or inhibiting effect on the cyanide, it will be desirable to subject the pulp to flotation separation before the addition of cyanide or other solvent. This will give a concentrate containing some of the gold values. Cyanide solution or other solvent will then be added to the remaining pulp, a precipitant being then added, and flotation carried out.

In many cases powdered wood charcoal will be the most satisfactory and economical precipitant. Nevertheless, other carbonaceous material, such as ground coke or graphite, may be employed. In addition, finely divided zinc or aluminum can be used. It has been found

that when using finely divided zinc as a precipitant, there is a tendency for partial re-solution of the gold during flotation, in which case an inert or reducing gas can be employed.

In the experiments mentioned, 3 and 5 lb. charcoal were added per ton slime and ore treated. Recoveries were 91 and 98 per cent. The Edquist process is being tried on a large scale in the Mount Morgan flotation plant.

CLEANUP FOR ZINC-GOLD-SILVER PRECIPITATE

Zinc Boxes.—As the operations in cleaning up the precipitate from cyanidation of gold and silver ores are so well known, little space need be devoted to them. Practice varies, but the work is relatively simple and causes little trouble. Where zinc shavings are used, generally the fine sludge is taken out of the compartments of the zinc box and later mixed with that from washing the long zinc. The sludge may be acid treated and then washed, dried, roasted and fluxed before melting, or it may be only dried and fluxed before melting. At some plants the entire contents of the zinc boxes are acid treated at every cleanup; but this is not advisable because it entails considerable labor, and as only new zinc is added to the boxes proper precipitation does not start so quickly as when at least half of the cells are filled with old zinc. When zinc shavings are used for precipitation, less than 60 per cent of the gold and not more than 75 per cent of the silver precipitated are recovered at any one cleanup, the remainder being returned with the old zinc to the boxes.

Filter Presses.—Cleanup of the filters used in the Merrill-Crowe process is much simpler and quicker. The precipitates from the filters are uniformly high in gold and silver and in many instances are weighed, fluxed and melted without preliminary treatment. In most plants the bullion is ready for shipment within 8 hr. after cleanup of the filters is begun.

On gold ores, the precipitates assay 45 to 85 per cent bullion, which in most plants is melted direct. Some operators prefer to give a muffle roast before melting, and in a few of the larger plants acid treatment is still used. In such plants, precipitates may be pumped from the precipitating tank direct to the acid-treatment tank, without intermediate handling.

In large plants the precipitate, either with or without acid treatment, may be melted with litharge, and the resultant lead

coupled, the bullion by this method being of much higher grade. In general, the cupellation method is to be recommended only where large amounts of gold bullion are produced.

In precipitating silver solutions, particularly when using Merrillite as a precipitant, the raw precipitate when taken from the filters contains 75 to 94 per cent pure silver, and this product is, of course, suitable for fluxing and direct melting without preliminary treatment. The moisture in the precipitate is sometimes reduced to 15 or 20 per cent before melting, but this is not necessary, particularly in the large stationary reverberatory furnaces used in the larger silver mills.

An important point in favor of this process and one that should appeal particularly to operators of customs works, is the ability to clean up and convert into bullion at any time all of the precipitated metals in the plant. Furthermore, the uniform ratio of precipitant to bullion and the fact that a complete cleanup is made make it possible to check accurately the bullion against both mill heads and residues and against solution assays, all of which makes the detection of theft a relatively simple matter.

MELTING

Cleanup at the Howey mine, Ontario, takes place three times a month. Flux consisting of 20 lb. soda ash, 24 lb. borax and 7 lb. silica sand is added to 100 lb. of the precipitate in making up the charge. Air used to force fuel oil from the supply tanks to the furnace is supplied by a Denver fire-clay blower operated at 3500 r.p.m. The Denver oil-fired tilting furnace is given a 160-lb. charge which, after melting for 2 hr., is poured into a conical-shaped cast-iron pot. When cooled, the buttons are separated from the slag, weighed and returned to the furnace for refining. This operation consists of melting in the furnace for 2 hr. with a flux made up of 7 lb. each of soda ash, borax and manganese dioxide. No skimming is done in the remelt, but when poured the molds are filled to overflowing until the gold appears at the surface.

TREATMENT OF HOLLINGER PRECIPITATES

In *Bul. I.M. and M.*, March, 1931 (see also *M.M.*, April, 1931) Matthew Scott describes in detail the production of fine gold and silver from Hollinger precipitate.

Zinc dust is used for precipitation.

ANALYSIS OF THE PRECIPITATE (DRY BASIS)

Gold.....	35.0%
Silver.....	7.2
Copper.....	1.4
Lead.....	9.3
Zinc.....	14.6
Iron.....	0.3
Alumina.....	0.5
Lime.....	11.7
Silica.....	0.7
Sulphur.....	4.4
	—
	85.1

By treating the precipitate with hydrochloric acid a residue of gold, silver and copper with only small amounts of the other elements is obtained.

ANALYSIS OF THE RESIDUE (DRY BASIS)

Gold.....	75.85%
Silver.....	15.37
Copper.....	2.27
Lead.....	0.15
Zinc.....	1.54
Iron and alumina.....	0.14
Lime.....	0.32
Silica.....	0.46
Sulphur.....	0.76
	—
	96.86

In the residue the gold and silver are present not as an alloy but as a mixture of finely divided material, and parting by sulphuric acid is thus feasible.

The gold residue is given a treatment with sodium acetate solution to remove traces of lead sulphate, which otherwise would make the gold bars brittle. It is then washed, dried, melted and run into bars over 992 fine in gold.

Silver is recovered from the sulphate liquor by precipitation with aluminum powder. It is washed, dried and melted with some soda and borax and obtained free from gold and about 985 fine.

FLUXING ZINC PRECIPITATE

As zinc precipitates will vary greatly in their chemical composition, no general rules may be laid down for their treatment for the production of fine bullion. Treatment may range from the acid process used at Hollinger (see above) to cupeling with litharge. Preceding melting the precipitate may or may not be roasted or treated with acid and after drying melted with a suitable flux.

The base metals to be slagged off may include, besides zinc, lead, copper, arsenic, antimony, bismuth, tellurium or selenium. The flux to be used naturally will depend upon the composition of the precipitate. Materials generally in use include borax, soda ash, silica, fluorspar, niter and manganese dioxide.

At the Wright-Hargreaves, Ontario, when direct melting in Rockwell furnaces, the average grade of precipitate was

	Per Cent
Gold.....	46.11
Silver.....	9.33
Copper.....	15.90
Zinc.....	9.31
Tellurium.....	6.29

The flux used for this precipitate consisted of:

	Per Cent
Borax glass.....	29.0
Nitrate soda.....	27.0
Silica.....	11.8
Fluorspar.....	7.2

TREATMENT OF PRECIPITATE

The raw precipitate treatment at the McIntyre-Porcupine has been 10 to 30 per cent, described by J. J. Denny in the *E. and M.J.* for November, 1933. The copper content of the ores has increased with depth and raw precipitate in 1936 carries 10 to 30 per cent copper, 8 to 9 per cent lead and about 25 per cent zinc. For the removal of the zinc and copper the precipitate is discharged from the presses into an acid treating tank partly filled with water to give a 3 to 1 dilution. For the removal of

zinc, sulphuric acid is added, and the charge agitated for 2 or 3 hr. Steam is admitted to the tank during both zinc and copper treatments. The tank is then filled with water, allowed to settle and the foul solution siphoned and run through the filter press. More acid is added, and agitation continued for 5 or 6 hr. after which the pulp is given a second wash to remove the bulk of the foul solution, which would cause trouble in the subsequent copper treatment. Sulphuric acid and manganese dioxide are then added in sufficient amount to attack the copper, the pulp agitated, washed and decanted as in the zinc treatment and the manganese dioxide treatment repeated as often as necessary to reduce the copper content to about 4 per cent. The pulp is then washed in warm water, filtered in a Perrin press, with paper as the filter medium, fluxed and melted in a Rockwell oil-fired tilting furnace for 3 hr. The crude bullion is remelted and refined by blowing air over the surface of the molten metal and skimming off the lead oxide and other impurities. The final bullion has a total fineness of 960 and contains 3.34 per cent copper, 0.48 per cent lead and 0.02 per cent zinc.

Cleanup at Randfontein Estates.—F. Wartenweiler gives us the following procedure at this great property where 600 stamps crush up to 12,000 tons of ore daily: The total extraction by sand and slime treatment of ore assaying 3.76 dwt. per ton is 95.4 per cent. Only 0.189 lb. cyanide is used per ton ore. Zinc dust precipitates 15,000 tons per day of 1.3 dwt. solution, the consumption of zinc being 0.05 lb. per ton ore. The cleanup is done every 14 days. Precipitate is hosed from the Merrill presses with a solution of potassium permanganate and weak cyanide. It is then pumped to vats and treated with 10 per cent sulphuric acid solution. Next, it is transferred to other vats and washed three times, the supernated liquor being decanted. Dehydrating is done in Johnson pressure filters. After calcining, the precipitate is charged into size-100 crucibles with a flux of borax, sand and manganese dioxide. Thirty of these crucibles are placed in a reverberatory furnace, where they remain 3 to 4 hr. The molten material is poured, and the bullion buttons remelted into bars weighing 1000 oz. These are sent to the refinery for a modified chlorine treatment. Slag is ground dry in a ball mill, and the product run over riffles. The tailings are sent to a smelter and run down with lead.

SAMPLING BULLION

Bullion may be sampled by dip while it is molten or by drilling or grooving bars. All the methods may be rendered reliable. At the Lake Shore, Kirkland Lake, Ontario, bars are drilled partly through from top and bottom at each end.

An example of good melting and sampling is that of the group of mines at Kolar, India: Sampling is by dip before pouring, and of the estimated 380,639.98 oz. of fine gold during one year the Bombay mint paid for 380,527.09 oz.

Dip sampling is preferred on the Rand as giving most accurate checks. The molten metal is well stirred and sampled with a graphite scoop; the gold is granulated in water and is then cleaned with hydrochloric acid before weighing for assay.

BULLION FROM SLAGS AND MATTE

Some metallurgists treat slag and matte at the mine; others prefer to send them to a smelter; but such by-products are difficult to sample and get satisfactory payment therefrom. Both of these materials contain shot bullion, and special crushing and concentrating will recover much of it; the residue is still gold bearing yet easier to sample at this stage. Then it may be sold to a smelter or fed gradually to the ball mills or tube mills, especially if the latter have a unit cell in the circuit to catch gold. If not too bulky, certain slags may be cupeled, and mattes may be melted with cyanide, a procedure that requires care against fumes.

BULLION PARTING

While bullion parting normally applies only to the large producers, the following information on Homestake practice is of some general interest and is taken from an article in *M. and M.*, December, 1935, by Nathaniel Herz.

The Miller or chlorine process of parting was adopted by the Homestake Mining Company in 1933, following the abandonment of the gold standard in the United States and the subsequent increase in the price of gold. Because 60 per cent of Homestake bullion comes from amalgamation and the balance from cyanide precipitate, this company had a choice of three methods by which the combined bullion could be treated, *viz.*, the Miller, electrolytic

and acid parting, and chose the first. The following analysis is representative of the bullions to be treated. The cyanide precipitate is treated in a reverberatory furnace and cupeled, which accounts for the presence of lead and antimony.

FINENESS OF BULLION

Parts per 1000	Recovered by	
	Amalgamation	Cyanidation
Gold.....	800 to 820	730 to 775
Silver	175 to 190	205 to 230
Copper.....	5 to 10	20 to 30
Lead.....	None	5 to 15
Iron.....	About $\frac{1}{2}$	None
Antimony.....	None	Trace to 5

The accompanying flow-sheet (Fig. 77) gives a good description of the parting operation by which all Homestake bullion has been treated since late in 1933. All refined bullion is shipped to the United States mint.

Silver is melted into bars averaging over 990 fine, although a process was developed whereby silver of standard purity, 999 fine, was made. After the process had been developed, new Treasury Department rules made it unnecessary to refine silver to this degree. With ordinary care, gold bars, as shipped, are about 994 fine.

ELECTROLYTIC REFINING

Few companies go to such trouble as Waihi and Real del Monte with their bullion. At the former, the zinc-box precipitate is treated with sulphuric acid, washed with hot water and dried to 25 per cent by suction. It is mixed with borax, 15 per cent by weight; soda, 8 per cent, and some sand and dried without stirring. Melting is done in No. 200 Morgan graphite crucibles heated by fuel oil. Bullion is sampled by drilling and is melted and cupeled with lead, yielding metal containing 15 per cent gold, 83 per cent silver and 2 per cent base. Slabs 8 by 10 by $\frac{1}{2}$ in. are poured. The gold is desilverized in Balbach cells and finally brought to 999 fineness in Wohlwill cells. The silver precipitate from the Balbach cells is washed and melted,

being 999 fine. Refinery charges are 6 pence (12 cts.) per ton of ore treated. In 1933, the total output was 69,185 oz. of fine gold and 413,983 oz. of fine silver.

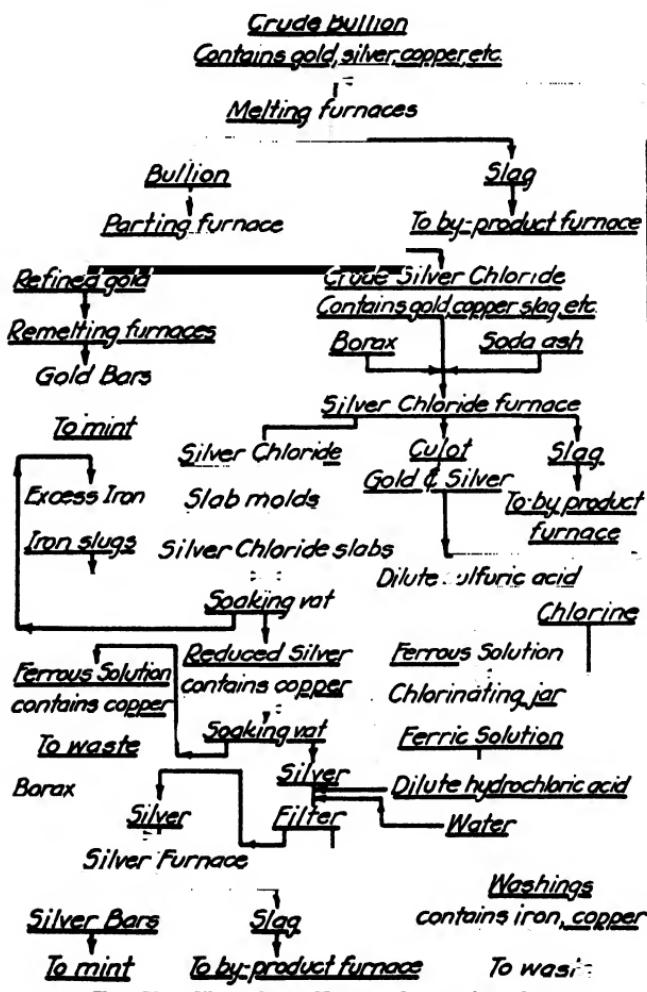


Fig. 77.—Flow-sheet, Homestake parting plant.

At Loreto, Pachuca, Mex., the melting and air refining of precipitates which assay 83 per cent silver and 0.46 per cent gold produce bullion 993 fine. Anodes weighing 22 lb. are parted in 200 electrolytic Thum-type cells, 52 by 24 by 9 in. The current is 3 volts and 160 amp. The electrolyte is copper nitrate

and silver nitrate, concentration being 75 grams silver and 95 grams copper per liter. The pure silver crystals are washed with distilled water, melted in a resistor-type electric furnace and cast into 1050-oz. bars, 999 fine.

The gold mud is treated with dilute nitric acid, melted and cast into anodes and refined in four Wohlwill electrolytic cells, 10½ by 10½ by 12 in. The current per cell is 1 volt and 110 amp. Electrolyte is gold chloride protected by hydrochloric acid. The cathodes are melted in an oil-fired furnace and cast into 1125-oz. bars, 999.8 fine.

GOLD RETAINED IN CIRCUIT

As soon as gold-bearing ore has been crushed or ground, some of the freed metal lodges in corners or behind mill liners and other places. Pulp that is splashed from launders and machines carries gold, which becomes lost temporarily. Pump and elevator sumps and boots retain gold. Stamp mills, especially those in which amalgamation is a feature, hold gold in the mortar boxes, copper plates and riffles. Splashings and spillage from the tables carry gold to the floor. Concentrators and classifiers hold gold. Ball-mill and tube-mill liners retain gold—on the Rand as much as 1000 oz. if grinding is done in water, and a tenth of this if done in solution. Some mills in Canada retain as much as \$25,000 in gold until cleaned out. Cyanide solutions that contain gold are absorbed by wood, and the metal is there held. Wood staves from treatment vats have been known to assay \$2000 per ton and were burned to save this gold. The precipitation and melting rooms always lose some metal. The dismantling and cleanup of many plants have yielded much gold to the owners or to others who have bought them. This temporary disappearance of gold, particularly in mills, is sometimes embarrassing to assayers and metallurgists who are at a loss to account for discrepancies. However, after a new plant has been running for several months, a balance is struck, and all gold is accounted for. Gold finds fewer places in which to lodge in the modern plant with its comparatively small amount of equipment and concrete floors which can be washed frequently.

Of the gold produced at the Pioneer mine, British Columbia, 4 per cent of the total is recovered from the ball mills and classifiers, when the mills are relined.

CHAPTER XI

PLANT CONTROL

Many factors enter into the control of operations in a cyanide or flotation plant. Those considered in this chapter include cyanides used and the effect of other agents on cyanide solution, the role of oxygen in cyanidation and the aeration and deaeration of solutions, the testing of solutions for strength and metal content, the regeneration of cyanide from waste solutions, lead salts and lead minerals in cyanidation, the effect of graphite, pH control in ore flotation, sampling ore and solution and sieve tests.

CYANIDE

The cyanides are compounds of the radical cyanogen CN. Their history covers more than two centuries, but that of the simple cyanides dates back about 90 years. (See *A Dictionary of Applied Chemistry*, 1921, by Edward Thorpe, pp. 437-475, Vol. 2; also *Die Technologie der Cyanverbindungen*, by Wilhelm Bertelsmann, 1906.) Prior to the introduction of the cyanidation of gold and silver ores in 1890, the world consumption of potassium cyanide was less than 100 tons a year. Now, including the cyanide used in casehardening, electroplating, flotation and fumigation, world consumption exceeds 30,000 tons of the calcium, sodium and potassium salts. Gold mines of the Rand consumed 8000 tons in 1933. Flotation processes in the United States use about 500 tons a year of the 12,000 tons for the whole country. Currently, throughout the world, approximately half of the cyanide used for ore treatment contains 98 per cent, and the remainder 49 per cent sodium cyanide equivalent. The principal American and Canadian sources of cyanide are Niagara Falls, N. Y., and Niagara Falls and Beachville, Ontario. In the United States the American Cyanamid Company makes the sodium salt and the Aero brand, and the Roessler and Hasslacher Chemicals Department of E. I. du Pont de Nemours Company makes the sodium salt. For many years the Cassel Cyanide Company, at one time known as the Cassel Gold Extraction

Company, a pioneer in cyanidation, had a cyanide-manufacturing plant at Glasgow, Scotland. This has been superseded by works at Billingham, County Durham, England. The company is a subsidiary of Imperial Chemical Industries, Limited. The well-known Cassel brand contains 97 to 98 per cent NaCN, equivalent to 129 to 130 per cent KCN.

Manufacture of Cyanide.—Sodium cyanide is a white, deliquescent, crystalline material easily soluble in water. The basic sources are alkalies or alkaline earths, atmospheric nitrogen and carbon.

In the United States it is derived (1) from sodamide which is produced from sodium and ammonia. The sodamide is heated with charcoal, and the resultant soda cyanamid is then heated with an excess of charcoal, resulting in the formation of sodium cyanide. This is the Castner-Roessler process which yields 96 to 98 per cent material.

In Europe, 90 to 92 per cent cyanide is manufactured (2) from destructive distillation of beet-sugar refuse, forming hydrogen cyanide. This is absorbed in caustic soda solution, from which the cyanide is obtained by evaporation.

Aero-brand cyanide is manufactured by fusing calcium cyanamid and salt in a continuous electric furnace, from which the product is tapped at regular intervals into a sump outside the furnace. The conversion is an equilibrium reaction, and practically all of the nitrogen is in cyanide form. By proper cooling of the melt, the equilibrium is "frozen" at the high-temperature equilibrium point. Samples of this cyanide, kept in sealed containers for several years, showed no change in cyanide content.

Types of Cyanide and Cyanide Consumption.—Of the two types of cyanide used in the cyanidation of precious-metal ores—sodium cyanide and Aero-brand cyanide—sodium cyanide is generally sold in the form of 5-lb. blocks or cakes and is packed in drums holding 200 lb. net.

Aero-brand cyanide is calcium cyanide containing 48 to 50 per cent of pure NaCN equivalent, the other half consisting chiefly of common salt and lime. It is manufactured in the form of black flakes and packed for shipment in zinc-coated iron drums. Its dark color is due to a small amount of graphitic carbon derived from the principal raw material cyanamid.

This brand of cyanide dissolves readily in water, leaving only a slight undissolved residue consisting chiefly of lime and graphitic carbon. This insoluble residue has no effect whatever on the dissolution of gold and silver and will not precipitate precious metals already in solution. It is, therefore, not necessary to remove it.

The recommended procedure for introducing cyanide into a cyanide-mill circuit is to provide a tank of sufficient size so that enough cyanide can be dissolved at one time to furnish the mill requirements for at least one shift. In the case of Aero-brand cyanide, a 10 per cent solution is recommended, which will then contain 5 per cent equivalent pure NaCN. It is advisable to aerate the solution with finely divided air (atomized) to eliminate the small amount of soluble sulphides that it contains. The addition of $\frac{1}{4}$ lb. lead acetate or pulverized litharge will speed up the desulphurizing operation. The tank for dissolving the cyanide should be provided with an agitating mechanism.

The strength of solution used in the cyanidation of precious-metal ores will vary with the type of ore and the precious-metal content. Cyanide plants treating gold ores, with little or no silver, rarely use a solution containing over 0.05 per cent NaCN equivalent. In the cyanidation of silver ores a stronger solution is necessary, and it is common practice to maintain a strength of 0.20 to 0.30 per cent NaCN. Mechanical and chemical losses of cyanide increase in direct proportion to the strength of solution; consequently, the cyanide content of solutions should be kept at the lowest strength consistent with maximum extraction.

The chemical consumption of cyanide, for a given ore, depends on the cyanide-consuming constituents present in the ore, the period of treatment and the strength of solutions. The mechanical loss of cyanide depends on the type of treatment employed. For gold ores the total cyanide consumption will average around $1\frac{1}{2}$ lb. NaCN per ton ore. In the case of silver ores the consumption is generally much greater, probably averaging over 2 lb. High-grade silver ores and concentrates may require as much as 10 lb. NaCN equivalent per ton.

(The foregoing notes were prepared by S. J. Swainson of the American Cyanamid Company, New York, which manufactures Aero cyanide. Reference may be made here to four papers on

cyanides and cyanidation presented to the Electrochemical Society in September, 1931: "Present Status and Uses of Cyanamid Process Cyanide," by G. H. Buchanan; "Cyanides in the Metallurgy of Gold and Silver," by E. M. Hamilton; "Cyanides in Metallurgy," by M. R. Thompson and "Physical and Mechanical Aspects of the Cyanide Process," by A. W. Allen.)

FACTORS IN DISSOLUTION OF GOLD AND SILVER

A careful study of the factors that influence the rate of dissolution of gold and silver in dilute cyanide solutions was recently undertaken by George Barsky, S. J. Swainson and Norman Hedley and published in *Trans. 112, A.I.M.E.*, 1934.

Cyanide Concentration.—The first of the series of experiments had to do with the effect of cyanide concentration on the rate of dissolution of gold and silver. In plant practice, the solution strength for gold approximates 0.05 per cent NaCN, or 1 lb. cyanide per ton solution. Stronger solutions do not seem to hasten the dissolution or improve the extraction; and as the chemical and mechanical loss of cyanide is much higher with strong solutions, obviously it is desirable to hold the solution at the minimum strength consistent with good extraction. The experiments covered the use of pure gold foil, solutions containing up to 0.50 per cent NaCN and a pH of 9 + but without alkali added. The maximum rate of dissolution of pure gold was reached at 0.05 per cent NaCN, corresponding to concentrations used in modern plants. The solubility of oxygen is practically unaffected by the concentration of cyanide.

A similar set of experiments was made on pure silver foil, in 0.01 to 0.50 per cent NaCN. The finding was a maximum rate of dissolution in 0.10 per cent NaCN.

A third lot of experiments was run on gold-silver alloys containing 79.8 per cent silver and 57.5 per cent gold. Sodium cyanide solutions of 0.10 per cent were used. Later, these were assayed and were found to contain gold and silver proportional to the composition of the alloys.

ALKALINITY VARIATIONS

A study was made of the effect of varying alkalinity on the rate of dissolution of gold in cyanide solutions. All tests were in cyanide solution of 0.10 per cent strength, and varying amounts

of lime water or sodium hydroxide were added. The rate of dissolution was greatly reduced at high pH values or high concentration of OH ions. As the curves plotted from the results were so different, further experiments were run to reveal this unexpected action of lime. It was found that lime had no appreciable influence on the solubility of oxygen in the cyanide solutions used, so calcium sulphate and calcium chloride were added. The former had a slight retarding effect on dissolution of the gold, and the other calcium compound had a more pronounced effect; but as it was determined that the reduction in rate of dissolution of gold caused by the addition of lime is due neither to lower solubility of oxygen nor to the presence of calcium ions, apparently both calcium and hydroxyl ion must be present to produce the full effect, as yet unexplained.

IMPURITIES IN CYANIDE SOLUTIONS

The following note is taken from *Rand Assay Practice*, 1932, edited by James Moir and G. H. Stanley:

That cyanide solutions in works practice do not remain pure in a chemical sense is to be expected, having in view their contact during treatment with the many contaminants found in gold or silver ore amenable to cyanidation. Impurities cause a chemical cyanide loss and at times detrimentally affect the extraction of the valuable metals sought. It is fortunate that in the practical application of the process, impurities do not usually accumulate to a prohibitive degree; this is to a large extent due to dilution of the solutions by the replacement by fresh water of the moisture leaving the plant with the residue.

With regard to visual signs of impurities, ferrocyanide colors solutions a brownish yellow of varying intensity. Other impurities are generally insufficient to create a characteristic color.

Impurities have their principal source in, and are traceable to the ore constituents, to contamination underground and to secondary reactions in the treatment at the surface. On the Witwatersrand the oxidation of the pyrite content of the blanket reef and wall rock is the chief source. The first place, in order of importance, may, therefore, be given to iron and sulphur in their many combinations and to reactions occurring during exposure in stopes through contact with water (containing dissolved oxygen) so freely applied in the course of dust-preventive measures. Pyrite and marcasite (FeS_2) and pyrrhotite (Fe_7S_8), where present, are converted (but to a relatively small extent) into soluble ferrous sulphate (FeSO_4), ferric sulphate, $\text{Fe}_2(\text{SO}_4)_3$, and free sulphuric acid (H_2SO_4), while colloidal sulphur may be set free. These, in their

turn, to avoid excessive corrosion of iron and steel equipment underground, are precipitated by the addition of neutralizing lime as hydrated oxides of iron, both in the ferric ($\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$) and ferrous ($\text{Fe}(\text{OHO})_2$) state, varying according to the degree of completeness of such neutralization and oxidation. The hydroxide in the ferrous state is particularly soluble in a cyanide solution. When these find their way to the cyanide-treatment plant, either with the ore or through the medium of mine make-up water, ferrocyanide ($\text{Na}_4\text{Fe}(\text{CN})_6$) and thiocyanate (NaCNS) are formed. This tendency to form acid ferrous salts and thus to destroy the oxygen necessary for gold solution is also latent in the ore undergoing cyanide treatment. As a measure of protection, an alkali, such as lime, is therefore provided and is available throughout; also corrective oxidizing treatment is applied.

An oxidation product of the sulphur, sodium thiosulphate ($\text{Na}_2\text{S}_2\text{O}_3$), plays a part. It is often present in the first effluent solution from sand treatment, and its general effect is, by decomposition in passing through the zinc-precipitation boxes, to form an insulating film of sulphide on the zinc which lowers precipitating efficiency. Complete oxidation of this compound to a sulphate appears difficult to attain in practice. Sodium sulphocyanide, (NaCNS) is present in practically all solutions in relatively small quantities. It has not been definitely proved to have a deleterious effect on gold extraction.

The alkaline sulphide, sodium sulphide (Na_2S), resulting from the reaction between the cyanide and ferrous sulphide (FeS), and generally supposed to have a retarding effect on gold and silver dissolution, is not often observed in solutions on the Witwatersrand, a fact which is due undoubtedly to its precipitation as zinc sulphide (ZnS) by the sodium zincocyanide ($\text{Na}_2\text{Zn}(\text{CN})_4$) present in all solutions where zinc is used as a precipitant and also to its oxidation to thiosulphate. In silver extraction its incidence is more pronounced, as the silver itself is often in direct combination with sulphide, as Ag_2S in the mineral argentite, pyrargyrite (Ag_3SbS_3), and in proustite (Ag_3AsS_3). Lead reagents, such as lead acetate, lead nitrate or lead oxide, are generally used as a safeguard, acting as precipitants of the sulphide.

Resulting from the abrasion and fracture of steel and iron in ore crushing and grinding, metallic iron is found in all mill pulps. Oxidation of this takes place to a certain extent through dissolved oxygen in the water and solutions employed and the aeration of the sand and slime incidental to the treatment process. Any ferrous oxide thus formed is attacked by cyanide solution and is a cyanide, since the resultant ferrocyanide is practically useless as a gold solvent.

With the use of zinc as the precious-metal precipitant, various reactions between this and cyanide take place, the principal compounds being zinc hydrate ($\text{Zn}(\text{OH})_2$), sodium zincocyanide ($\text{Na}_2\text{Zn}(\text{CN})_4$) and

sodium zinc ferrocyanides ($\text{Na}_2\text{ZnFe}(\text{CN})_6$)² and ($\text{Na}_2\text{Zn}_2\text{Fe}_2(\text{CN})_{12}$). Sodium zincocyanide is a solvent of gold, as shown by Julian and Smart. The amount of zinc dissolved is considerable, but its retention in solution is not cumulative, as it is precipitated by reaction with the sulphide constituents of the ore in the ordinary course of treatment and by the ferrocyanide. Sodium cyanide is regenerated in the same reaction, the cyanide loss, therefore, being much less than at first would be expected.

Calcium is introduced in the form of lime (CaO) for the purpose of providing a neutralizing agent. Its use results in the formation of calcium carbonate (CaCO_3) and of calcium sulphate (CaSO_4). On the Witwatersrand, the treated mine water used as water supply probably introduces the greater portion of the CaSO_4 content of the solutions.

With favorable temperature or saturation conditions it crystallizes out over the entire plant, including the interior of pipes, and may become troublesome. As a physical obstacle and an insulator of zinc, it may be considered objectionable. The use of sufficiently clean water is the best preventive. Sodium carbonate is sometimes used to remove it as precipitated calcium carbonate. Magnesia is introduced to a small extent from underground sources, and finds its way into the solutions as magnesium sulphate (MgSO_4) and magnesium carbonate (MgCO_3).

Gelatinous silica is often found in cyanide solutions. Its effect in practice is more physical than chemical, in clogging filters and extractor boxes. Its source is the action of acid mine waters on the ore constituents. Silica may also be introduced in the form of calcium silicate as an impurity in lime.

Organic matter is a common source of impurities in cyanide solutions, its reducing effect being notorious. It is usually regarded as having its origin in mine timber, sewage and sacking, coming from underground and from the surface in the form of vegetal matter and sewage contamination of water used in milling. In the self-decomposition of cyanide solution, organic compounds such as formates are formed. Prevention is the soundest remedy. Failing this, oxidation by means of chlorine oxidizers has proved efficacious where these can be applied directly or in a separate circuit before cyanide treatment. When strong oxidizers are used on cyanide solutions, free cyanide will be lost by conversion to cyanate.

In cyaniding ores containing copper minerals, it is found that the carbonate, oxide and sulphate minerals particularly are attacked by cyanide with avidity, causing a heavy cyanide consumption by the formation of cuprosocyanide ($\text{KCu}(\text{CN})_2$). In practice this is minimized by the use of extremely weak solutions. Provided that the copper content of the ore is not excessive, it is found that copper does not accumulate in the solution, as it is constantly being precipitated by sodium sulphide.

THE ROLE OF OXYGEN IN CYANIDATION

Although, originally, MacArthur doubted that oxygen had any effect on the cyanide-gold reaction in the dissolving process, the important part that it does play is now generally recognized. As others have done, we give the equation to aid in an understanding of what follows:



Oxygen is therefore necessary, and as much as can be supplied to solutions, cheaply, in the form of air, is desirable. Oxidizers have been tried, but they are expensive and unsatisfactory. The

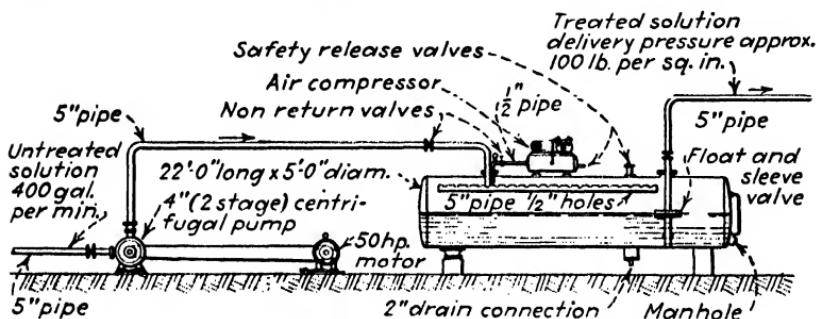


FIG. 78.—Cylinder for aerating cyanide solutions.

oxygen content of solutions is lowered by such reducing substances as ferrous compounds dissolved from the ore being treated or by other means. In general, simple exposure to the atmosphere replenishes the lack of oxygen; but from a practical standpoint appreciable air must be forced into the pulp undergoing treatment to maintain a satisfactory oxygen content.

Aerating or Oxygenating Cyanide Solutions.—A new process for aerating or oxygenating cyanide solutions was announced by T. K. Prentice in the *Jour., C.M. and M.S.S.A.*, February, 1934 (see Fig. 78). It immediately attracted attention, and the article was reprinted in part by United States and Australian technical journals. The process received practical plant trials at the Nourse mine on the Rand before it was made public. These were mainly on sand which is leached at the mine. Gold extraction was slightly higher when aerated solutions were used, and consumption of cyanide was a third less than in regular treatment. The oxygen content of solutions at the Nourse

mine for dissolving gold ranged from $2\frac{1}{2}$ to $5\frac{1}{2}$ milligrams per liter and averaged 4 milligrams per liter. At one time it fell to 1 milligram per liter, and a series of high residues resulted. Six milligrams per liter is considered desirable. The oxygen content of circuit solutions in 16 plants on the Rand averages 4.5 milligrams per liter.

In brief, the oxygenating process is as follows, with reference to Fig. 78, which is the patented plant-scale equipment: Cyanide

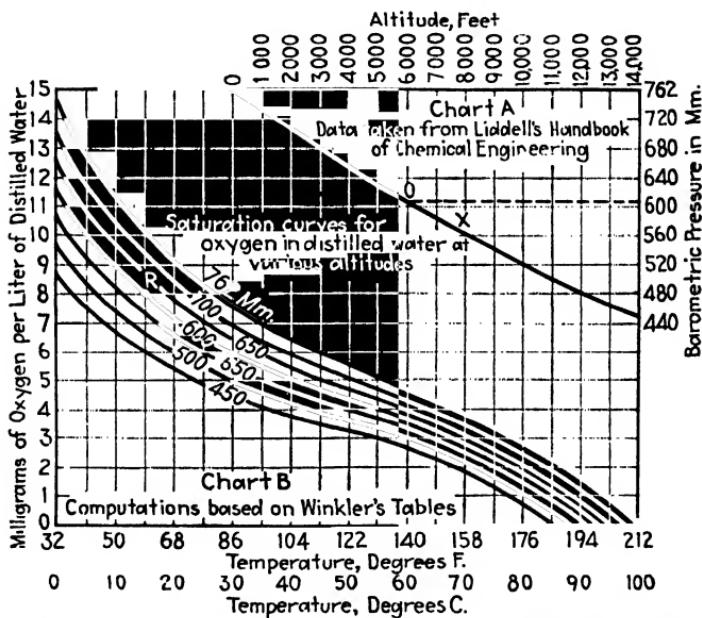


FIG. 79.—Curves (A) for determining barometric pressure at various altitudes and (B) for determining standard oxygen-saturation values at various temperatures and pressures.

solution is drawn from a stock tank and pumped into the drum or cylinder shown, first passing through the pipe with $\frac{1}{2}$ -in. holes to form a spray. Air at 100-lb. pressure is generated by the motor-driven (3 hp.) compressor atop the cylinder. The aerated solution leaves at the rate of 2 tons per min., the balanced float valve shown, attached to the discharge pipe, regulating the level of solution in the cylinder. The solution is milky white, owing to the disseminated air, but it clears in a few minutes. At this stage it may carry 6 milligrams of oxygen per liter and retains

within 2 milligrams of this amount for 22 hr., which is long enough for the solution to be effective during the first stage of leaching.

Aeration or oxygenation of sand, slime and cyanide solutions has been practiced during the several decades of cyanidation—purposely and incidentally—but not until recently has it been given the careful study that it deserves. Every time a pulp or solution is stirred or transferred it absorbs some oxygen, but special methods or devices have been developed to entrain air in pulps and solutions. Aeration in the Dorr agitator is obtained from the compressed air used for circulating the pulp through the revolving central lift column and also from the atmosphere when the elevated pulp is redistributed over the pulp surface in a series of small streams from the distributing launders. Additional compressed-air jets are sometimes attached to the revolving arms for greater aeration. In the Pachuca agitator compressed air used for circulating pulp through the stationary lift pipe is the only source of oxygen. The Devereux and other types of mesh-propeller agitators rely on the vortex created to entrain air.

The Turbo- and Wallace-type agitators, both of which thoroughly incorporate air in the pulp through the action of their impellers, are used in several of the Kirkland Lake mills for agitation in small tanks and are also used in the top of and near the surface of Dorr agitators to increase normal aeration.

Aeration during treatment of silver ores is essential. At Pachuca, Mexico, air at 18 to 35 lb. pressure is used for 70 to 73 hr. for the combined purpose of mechanical circulation and oxygenation. Oxygen consumption is 210 grams for the dissolution of 292 grams of silver per ton of ore or about 9 oz per ton. This is 9.75 times the amount called for by the well-known equation.

Oxygen Content of Cyanide Solutions.—Two methods for determining the oxygen content of cyanide solutions are offered as being simple and accurate—that of H. A. White, as described in the *Jour. C.M. and M.S.S.A.*, June, 1918; and that of A. J. Weinig and M. W. Bowen, described in *Trans. 71, A.I.M.E.* 1925.

White's method is a colorimetric one, depending on the degree of coloration imparted to a solution of pyrogallic acid in the presence of caustic soda. Weinig and Bowen's method, a modification of that of Schutzenberger, depends on the reducing action

of a sodium hydrosulphite solution on a solution of indigo blue (indigotin disulphonate).

White Method.—This method was developed by H. A. White, consulting metallurgist for the Union Corporation of Johannesburg, and is based upon the color imparted to an alkaline solution on the addition of pyrogallic acid, the "pyro" used in photographic work.

APPARATUS REQUIRED

- 1 dozen 250-cc glass-stoppered bottles.
- 1 burette.

CHEMICALS REQUIRED

- Sodium hydrate (NaOH) solution, 2N (80 grams per liter).
- Pyrogallic acid or pyro (the crystalline salt is preferable to the powder).
- Brown dye (Diamond brown or caramel).

PREPARATION OF STANDARD COLORS

Saturate a quantity of ordinary tap water with oxygen by passing air through it for an hour. Then stand for another hour to remove bubbles of occluded air. To one of the 250-cc bottles containing this water add about $\frac{1}{10}$ gram of pyro and 1 cc 2N-NaOH. The pyro crystals must immediately sink below the surface. Then insert a glass stopper with a twisting motion to exclude any small air bubbles. After the soda solution is added, the bottle should be filled to within $\frac{1}{4}$ in. of the top, so that the stopper may be inserted at a slight angle. Shake the bottle until all the pyro is dissolved.

The water will take on a reddish-brown color corresponding to the oxygen content of oxygen-saturated water at the existing temperature and pressure. The percentage oxygen content of water, saturated under any given set of conditions, may be calculated from the chart of Weinig and Bowen (Fig. 79). This color is then matched with a water solution of Diamond dye or caramel. A small amount of chromate of potash will sometimes assist in obtaining the exact shade.

Assume that under the existing conditions the saturation point of water corresponds to 8 milligrams oxygen per liter. Then if standard bottles are made up containing 1 part color corresponding to saturation and 7 parts water, this lighter color will correspond to 1 milligram oxygen per liter, and equal parts of

standard color and water will correspond to 4 milligrams oxygen. In the same way make up a series of eight bottles, colors in which represent oxygen contents of from 1 to 8 milligrams oxygen per liter.

TESTING THE SOLUTION

Fill one of the 250-cc bottles with the solution to be tested. Use a rubber tube reaching to the bottom of the bottle, and avoid all agitation. A drop or two of kerosene oil in the bottle will film the solution and still further prevent absorption of oxygen. Fill the bottle nearly full, then add $\frac{1}{10}$ gram pyro and 1 cc 2N NaOH, and stopper instantly, taking care that no air bubble is left under the stopper. Shake well, and compare with standard colors.

Some solutions, particularly those resulting from the cyanidation of silver ores, show fading colors and become cloudy on the addition of pyro and soda, according to E. M. Hamilton in *E. and M.J.*, July 17, 1920. In such cases a better comparison is made after the solutions stand a definite time, say 3 to 6 min. Also, in such solutions a better standard color is obtained by making up with a regular plant solution to which is added the usual amount of soda and pyro. Then, after standing 3 to 6 min., this color is matched with the dye or caramel, and the fractional standards prepared as usual. A small amount of solid pigment such as yellow ocher, added to the dye or caramel solution, will match the precipitate which sometimes forms. With such solutions it is better to make up a fresh set of standards whenever oxygen tests are to be made.

Weinig-Bowen Method.—The Weinig-Bowen method determines oxygen accurately to tenths of a milligram per liter of solution or 1 part oxygen in 10,000,000 parts of a solution on a 250-cc solution sample, with a proportionately greater degree of accuracy on larger samples.

Reasonably clear mill solutions are best sampled by siphoning them through a rubber tube and glass tube into Winchester acid bottles. Pulps should be settled, and the clear liquor siphoned off.

A convenient quantity of standard sodium hydrosulphite solution is made up as follows: Fill a $2\frac{1}{2}$ -liter acid bottle with distilled water, preferably fresh. Dissolve in it 5 grams caustic

soda, and then add 5 grams sodium hydrosulphite. Place a layer of kerosene over the solution. Then siphon the liquor into bottle *b* of Fig. 80. This solution deteriorates rapidly if exposed to the air, so, as shown in the insert *a* of the sketch, the cork (not rubber) is run in with shellac.

The indicator, indigotin disulphonate, is made up as follows: Place in a casserole 7 grams indigotin, and add 30 cc concentrated sulphuric acid. Place over a water bath, and heat to 90°C. for

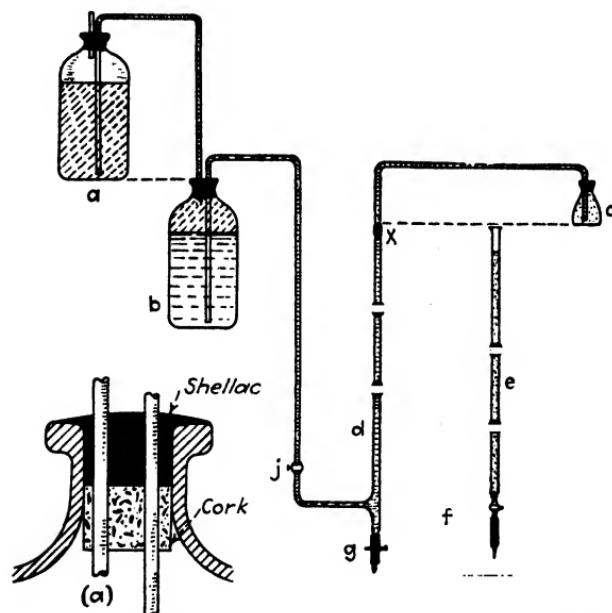


FIG. 80.—Apparatus for determining oxygen in cyanide solutions.

1½ hr. or until all lumps disappear. Then dilute to 2 liters with distilled water. Neutralize the acidity by adding powdered limestone, a little at a time, allowing it to stand a few minutes between additions, until all action has ceased. Filter without washing, place in a corked bottle and use as required. It is convenient to dilute this solution so that 1 cc of the indicator is equivalent to 0.25 milligram oxygen per liter solution. This will indicate 1 gram per liter when a 250-cc solution sample is taken for titration. This indicator does not deteriorate and may be kept in a well-stoppered bottle.

Figure 80 shows the apparatus for this test as follows: Two $2\frac{1}{2}$ -liter acid bottles *a* and *b*; a 250-cc flask *c*; a 50-cc burette *d* with side connection; a common burette *e*; a clamp stand *f* to hold two burettes; a 400-cc beaker with 250-cc point scratched on it; a glass stirring rod; $\frac{3}{16}$ -in. glass or lead tubing and rubber tubing for connections; a pinchcock *g* for bottom of the rubber connection on the burette that contains standard hydrosulphite solution and a container for kerosene to be used in the procedure. When setting up the apparatus the relative positions of the parts shown should be closely observed.

The bottles are filled as follows: Remove the connection *x*, and place a cork stopper in the top of the burette so that no solution can overflow. Place a bottle containing $2\frac{1}{2}$ liters of kerosene so that its bottom is above the top of bottle *a*, and connect this bottle to the bottom of burette *d* with a siphon. Open pinchcock *g* and stopcock *j*, and allow kerosene to siphon into bottle *b* until filled. Replace the bottle that contained kerosene by a bottle containing the standard solution of hydrosulphite. This solution should always be covered by a layer of kerosene; siphon the standard solution into bottle *b*, the kerosene being forced from bottle *b* over into bottle *a* automatically. As soon as the hydrosulphite solution has reached to within 1 or 2 in. of the top of bottle *b*, close both the pinchcock *g* and stopcock *j*. After flask *c* has been nearly filled with kerosene, place connection *x* in top of burette *d*, and seal with dry shellac dissolved in alcohol. Open stopcock *j*, keeping pinchcock *g* closed, and allow the standard solution to pass into burette *d* until it just enters flask *c*; then close stopcock *j*, open pinchcock *g*, and allow the standard solution to drain completely; its action as a siphon will draw the kerosene over into burette *d*. The standard solution is now drained off to eliminate any possibility of its being exposed to air and to give it a cover of kerosene in burette *d*. Close pinchcock *g*, open stopcock *j* and allow burette *d* to fill to the zero mark. The layer of kerosene prevents admission of air during this procedure. Then the apparatus is ready for use. Fill burette *e* with the indicator, and place a glass cover over the top to prevent evaporation.

Routine titrations are as follows: The solution sample is siphoned over beneath the kerosene into the 400-cc beaker to the 250-cc mark. Alkalinity is neutralized with dilute

sulphuric acid, 1 cc or less of indigotin disulphonate is added as indicator and titration is completed with the hydrosulphite. Then the necessary connection is made for the indicator, and the result is converted into milligrams of oxygen per liter of solution or percentage saturation, as may be desired. The end point in clear solutions is a slight yellow, but with others it may be white to gray, especially if certain salts are contained. The kerosene may be used several times by pouring the contents of the beaker into a large bottle, after titration, then siphoning off the kerosene for re-use after enough has accumulated. General circulating-plant cyanide solutions have 7 to 75 per cent maximum oxygen-saturation. A number of precautions must be taken, but these will probably suggest themselves.

Altitude-pressure and Standard Saturation Curves.—Chart A (Fig. 79) is used to determine barometric pressure at various altitudes; chart B is used to determine standard saturation values for various temperatures and pressures. In Chart A, altitudes, in feet, are plotted on the horizontal axis; and pressures, in millimeters, on the vertical axis. To find the pressure corresponding to a certain altitude, follow the elevation line downward to its intersection with the curve x , then horizontally to the right, and read the pressure. For example, if the elevation is 6000 ft., the 6000-ft. line is followed to its intersection o with the curve x ; then the corresponding pressure, 607 mm, is obtained from the right-hand side of the chart.

In chart B, temperatures are plotted on the horizontal axis, and the amount of oxygen, in milligrams per liter of distilled water, is plotted on the vertical axis; various pressure curves also are plotted, as shown. To find the saturation value for a certain temperature and pressure, follow the temperature line upward until the point corresponding to a given pressure is reached, then follow horizontally across to the left-hand side of the chart, and read off the amount, in milligrams, of oxygen per liter of solution. For example, to determine the amount of oxygen in a solution having a temperature of 59°F., at an elevation of 6000 ft.: The pressure is 607 mm; at this elevation, as found in chart A, therefore, the 607-mm pressure curve must be used; *i.e.*, it is necessary to interpolate between the 600- and the 650-mm curves. The 59°F. line is followed to its intersection R with the 607-mm curve, then from the left-hand side of the chart is read off 8 milligrams

of oxygen per liter. The same procedure is used for the various pressures, using the curve corresponding to the particular pressure. For any particular plant, a solubility curve should be plotted based on the altitude at the plant.

Oxygen in Mill Solutions.—Results of experiments conducted in the South Kalgoorlie dry-crushing, all-roasting and cyaniding plant at Kalgoorlie on dissolved oxygen in mill solutions were given by C. W. Brown in *C.E. and M.R.*, September, 1934. Pulp agitation is done in vats with ordinary stirrers, but considerable compressed air is also introduced. White's method of oxygen determination was unsuitable, but Weinig and Bowen's was found to be satisfactory. The average altitude in the plant is 1271 ft. above sea level.

The average value of dissolved oxygen just prior to filter pressing the slime was 25.05 per cent saturation, and the average of many determinations in which the agitators were sampled just prior to filter pressing was 28.0 per cent, with a range of 4.2 to 57.5 per cent. There seemed to be no definite relation between the amount of oxygen in solution and solubility of the gold. The oxygen content of plant solutions was as follows:

OXYGEN SATURATION IN SOUTH KALGURLI SOLUTIONS

Place	Oxygen, per cent	Tempera- ture, degrees
Head solution tank.....	70 to 85	40
Grinding pans.....	2 to 5	57 to 65
Thickeners.....	16 to 20	54
Agitators.....	5 to 57	45
Filter presses:		
Pregnant.....	5 to 57	45
Washes.....	50 to 65	34 to 37
Merrill-Crowe unit:		
Feed.....	Zero to trace	
Discharge.....	60 to 80	34

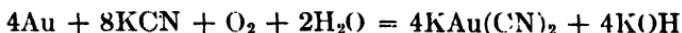
When cyanide solution is in contact with ore the oxygen content is reduced, whereas when it leaves the ore at any point it starts at once to dissolve oxygen.

Experiments to determine what are the oxygen retardants or absorbents revealed that among the solubles are sulphites and

hydrogen, and among the insolubles are metallic iron and ferrous sulphide.

THE SOLUTION OF GOLD

"The Physics of Gold-Solution," by H. A. White, in *Jour. C.M. and M.S. S.A.*, July, 1934, is a highly technical dissertation, wholly confined to the surface reaction between metal and cyanide solution. White's experiments were based on the hypothesis that the rate of solution of gold in cyanide solutions is mainly dependent upon the presence of oxygen. The chemical reaction between gold and cyanide may be expressed as



According to this equation, 1 milligram gold requires 0.0406 milligram oxygen, and this corresponds with 5.80 cc solution at 7 milligrams per liter. At an oxygen concentration of 8 milligrams per liter the corresponding KCN strength is 0.01302 per cent or 0.00980 per cent NaCN. It is found, however, that gold dissolves at the maximum rate if the solution contains 0.027 per cent KCN, equal to 0.020 per cent NaCN, and if it is saturated with oxygen. This difference is due to the slower diffusion rate of the cyanide and to incomplete ionization and hydrolysis for which an allowance of 10 per cent must be made.

When the cyanide concentration is lower than the optimum, its diffusion rate will be the determining factor in a saturated oxygen solution, but a greater concentration can only hinder rate of oxygen diffusion as well as reduce its solubility. On the other hand, the presence of excess oxygen, either by means of increased pressure or in the presence of oxidizing agents which attack the cyanide slowly enough, will raise the optimum cyanide concentration and the maximum rate of gold solution. If less than saturated with oxygen, as is frequently the case in working solutions, the cyanide strength could be correspondingly reduced, and, in any case, the rate of attack on the gold diminished.

It is known that the rate of gold solution is increased by contact with zinc, iron or carbon, and this may be attributed in the last resort to the extension of surface to which the oxygen may diffuse and likewise involves an increase in the optimum cyanide concentration.

Four methods of treatment were employed in the experiments—slime treatment by agitation; sand treatment by percolation; gold plates hung in still solution; gold plates hung in moving solution. The optimum cyanide strength was used, and the oxygen concentration was 7 milligrams per liter. These tests, with the physics involved, are given in detail.

CONDITIONING ORE PULP

Conditioning of an ore pulp is understood to mean its preparation for subsequent treatment by cyanidation or by flotation. It is an important step, and its importance is gaining recognition.

Conditioning is more effectively done in so-called contact tanks than in ordinary agitation tanks. Generally, a conditioning tank is a small agitator into which the pulp is pumped just after lime, cyanide or flotation reagents have been added to it, the chief purpose being a thorough mixing. The period of contact is determined in the laboratory and depends upon the nature of the ore, the physical condition of the pulp and the amount and kind of reagents. The size of the conditioning tank is dependent upon the period of contact and the tonnage to be treated. Broadly speaking, any preparatory machine may be spoken of as a "conditioner." Grinding is merely conditioning the ore for cyanidation proper. If fine grinding is essential, then ball mills, tube mills, and classifiers become important conditioners. The thickening of pulp may be termed conditioning, for subsequent cyaniding operations. Air lifts in agitators are likewise conditioners. This idea that the practical purpose of every machine is to condition the ore for the subsequent machine is advanced by L. E. Djingheuzian of the Lake Shore mines, Ontario.

Cyanidation.—Conditioning operations at the Dome plant in Ontario comprise grinding in water to produce a pulp 83 per cent through 200 mesh which is passed over corduroy. Tailing from the corduroy tables, after the addition of lime, is aerated in four 14- by 40-ft. Pachuca tanks. The pulp is then treated with cyanide and agitated in a series of 10 Pachuca tanks.

At the Willow Creek plant in Alaska only the concentrates are conditioned by grinding in lime water to 98 per cent through 200 mesh. After aeration by means of air lift, the pulp is agitated with cyanide solution in three 6- by 8-ft. Devereux agitators.

Flotation.—The conditioning of ore pulp before flotation is of importance in treating gold and silver ores as well as base-metal ores. Lack of uniform results in a flotation mill can often be attributed to one or all of the following conditions: (1) Lack of proper time for chemical and physical reaction; (2) incomplete mixing of reagents; (3) fluctuation in the mill feed, causing unequal pulp flow and lack of uniformity in reagent content or (4) excessive consumption of reagents due to improper conditioning.

The primary purpose of a conditioner, in flotation, is to mix the reagents thoroughly with the pulp before flotation and allow time for the chemical and physical action to take place. These objectives, of mixing and of completing the chemical and physical reactions, can be attained with some reagents by adding them ahead of the grinding unit. Excessive oxygen beaten into the pulp during grinding precludes the use of certain reagents because oxidation makes them ineffective. Other reagents, because of their frothing properties, affect classification and cannot be used in the grinding units. By using a conditioning tank in the circuit, however, the operator has accurate control of the time of agitation, which can be varied for different ores and reagents. Complete mixing of the reagents is also assured.

In addition to mixing the reagents uniformly with the pulp, the conditioning tank serves as a source of uniform feed to the flotation cells, acting as an equalizing tank or stabilizer. Without this tank in the circuit, the flow of feed from the grinding circuit is likely to be uneven, and the reagent content irregular, thereby interfering with proper operation of the flotation cells. A uniform feed naturally improves the operating efficiency of a flotation unit.

CYANIDE RECOVERY OR REGENERATION OF CYANIDE

CYANIDE CONSUMPTION AND REGENERATION

With regard to regeneration of cyanide, J. E. Clennell said in *The Cyanide Handbook*:

Since the main cause of cyanide consumption is the formation of soluble double cyanide or complex cyanogen compounds of the base metals, and since solutions highly charged with such compounds are more or less inefficient as solvents of gold and silver, it has been suggested that the cyanogen in such liquor might be recovered in the form of simple alkali cyanides, by treatment with suitable chemicals. This

has been carried out in practice in some cases, but generally speaking the cost of chemicals, power, labor and other charges required for such treatment outweighs the advantage gained by it.

The latter part of the foregoing statement may be questioned by some, but although the few regeneration plants at work report a saving in cyanide and an improvement in the treatment, methods for the recovery of cyanide have not been generally adopted. A. W. Allen believes that much will be accomplished when regeneration of cyanide is economically applicable to the low-strength solutions used in the treatment of gold ores.

Under normal operating conditions the mechanical loss of cyanide discharged with the tailing is an important factor in cyanide consumption. This loss is because of imperfect washing, which in turn is the result of limiting the water wash to the amount of water necessary to maintain a balance of plant solution. This difficulty can be practically eliminated by using solution, from which the cyanide has been removed, to extend the washing period, according to W. E. Crawford of Fresnillo, Mexico.

In Handbook of Ore Dressing, by A. L. Taggart, R. C. Canby said:

Precipitation of gold and silver from cyanide solution by means of zinc or aluminum results in regeneration of cyanide, probably not in the form of alkali cyanide as originally added but in a form in which an equivalent amount of effective cyanide ion is present.

The common method for regeneration of cyanide is by acidulation of the solutions. All or part of the cyanogen is converted thereby into hydrogen cyanide, which is fixed by an alkali (generally lime) and returned to the cyaniding system. This is the Mills-Crowe process, the principle of which is described by C. W. Lawr in *T.P. 208, A.I.M.E.*, 1929, in which he also gives a selected list of 38 references to regeneration:

PRINCIPLE OF CYANIDE REGENERATION

The solution, be it a weak wash or a foul solution, is made acid by bringing it into contact with sulphur dioxide. The acidified solution is then transferred to a closed tank in which air and solution are brought into intimate contact. The air leaving the tank charged with hydrogen cyanide is then passed to another tank in which it is mixed with an alkaline solution, the latter absorbing the HCN and leaving the accom-

panying air clean for re-use in removing more HCN from the acidified solution.

The extent to which the acidified solution will become impoverished of its cyanide will depend upon the acidity, the amount of air brought into contact with the solution and the quantity of residual HCN left in the air after the latter has passed the absorbing apparatus. (The system is closed so that the same air is used repeatedly.)

Of course the amount of air required will depend on how efficiently it is utilized, but where other conditions are equal it may be stated that the amount of cyanide removed from a given volume of solution increases with an increase in air; and, further, an increase of air will do more good, or a decrease will cause poorer results than almost any other change that could be made in the plant. (By increase in air is meant an increase in the velocity of that being circulated.)

The impoverished acid solution may be wasted or used as a water wash on the filters, either before or after filtering, depending on whether the solution contains enough silver or copper to pay for its removal. If the spent solution does not contain any silver, but much copper, it is doubtful whether it could be used as a filter wash before filtering, because the slimy nature of the precipitate would affect adversely the filter leaching rate.

The Mills-Crowe cyanide-recovery process is applicable only in plants of rather large tonnage and best on solutions resulting from the treatment of silver ores or those containing appreciable amounts of copper. At the present time, seven regeneration plants are operating, with an aggregate capacity of approximately 10,000 tons of solution daily. Four of these are briefly described; the other three are at the San Luis mill, San Dimas, Durango, Mexico; South American Development, in Ecuador and Hudson Bay at Flin Flon, Manitoba, Canada. Primarily this process effects a reduction in net cyanide consumption by recovering the cyanogen which is ordinarily wasted in mill residues.

Cyanogen in mill solutions exists chiefly as free alkali cyanide, as zinc and copper double cyanides and as sulphocyanide and ferrocyanides. From the free cyanides and zinc double cyanide, substantially complete recovery of the cyanogen is easily effected. Part of the cyanogen combined with copper is readily recovered; regeneration of the remainder, with the cyanogen existing as sulphocyanide and ferrocyanides, requires special treatment which well may be justified in large plants or in plants where unusually strong solutions are employed.

From Silver-ore Treatment.—W. E. Crawford, in *Trans. 112, A.I.M.E.*, gives the following description of treatment at Fresnillo:

For details of the process the reader is referred to C. W. Lawr's excellent paper. The plant that he describes is very similar to the installation at Fresnillo, the principal difference between the two plants being that the Fresnillo plant removes the cyanide from the barren solution, using sulphur dioxide gas obtained by the roasting of pyrite concentrate from the Fresnillo Company's concentrator, whereas the plant of the Compañía de Santa Gertrudis, S. A., at Pachuca, removes it from an extra water wash after the plant balance has been satisfied, using sulphur dioxide gas obtained by burning 98 per cent sulphur.

One point has been developed in the Fresnillo practice of using treated barren solution; namely, that it effects an attractive saving in cyanide, but, at the same time, the slightly acid solution has a precipitating effect on the silver in the filter cake—as noted under filter practice. Also, the process introduces another adverse factor, because some sulphur dioxide is absorbed with the hydrocyanic acid gas by the mill solution in the absorber towers, and this sulphur dioxide gas forms lime salts that absorb the oxygen in solution, which must be replaced by mechanical means in order to obtain the best metallurgical results, especially when treating gold ores. Wherever headroom is available, perforated screens are used under pump discharges to obtain an intimate contact between the solution and the air. Experiments are now being conducted with a 27-in. turbomixer in the mill-solution storage tank, to increase the oxygen content. A preliminary test of one hour raised the oxygen in 380 tons of mill solution from 68.5 to 89 per cent of saturation; in practice this one mixer raises the oxygen content of the mill solution 6 per cent.

The operation of the cyanide recovery plant is very simple and requires only one man per shift under the direction of the cyanide-solution operator. During a normal month (for instance, July, 1934), 75,000 tons of solution was treated. The heads titrated 0.054 free cyanide, 0.073 per cent total KCN and 0.038 per cent CaO. The final treated solution titrates 0.006 per cent KCN.

At Pachuca, Mexico.—a. The ores mined at Santa Gertrudis are mainly silver bearing, but they yield also some gold and a little copper. Strong cyanide solutions are used, and the mechanical loss at the tailings filter is large, considering a daily discharge of 1600 tons of slime with a liquid-to-solid ratio of 2 to 1. The cyanide-recovery plant has a maximum capacity of 2000 tons of solution in 24 hr.

The sulphur dioxide gas used in the process is made in a rotary sulphur burner. It is brought into contact with the cyanide solutions in an acidifier. The HCN formed in the acidifier remains in the acid solution until removed in the dispersers. This gas is dissolved in the solution, and although it can be removed fairly easily, it is fixed sufficiently in the acidified solution, so that upon passage through a weir box 6 ft. in length open to the atmosphere under normal conditions no loss of HCN can be detected by silver nitrate titration of the solution as it enters and leaves the weir. At times, an odor of HCN is noticeable, indicating a slight loss.

The HCN is removed from the acidified solution by bringing a large volume of rapidly moving air into contact with the acidified solution spread over a large surface. Solution-surface exposure is obtained by grids and spraying devices. At least 15 cu. ft. air per min. per ton solution treated in 24 hr. is required.

The HCN removed from the acidified solution then is absorbed by an alkaline plant solution in horizontal absorbing towers in which the solution and the air containing the gas are brought into close contact.

If silver and copper are present in the filter wash that has been deprived of its cyanide, they will be thrown out as precipitates. The treated solution may be filtered, and the slimy material dried and sold to a smelter. Gold does not appear to be precipitated.

Careful control and testing are essential in carrying out the foregoing process. In 1929 the Santa Gertrudis plant was recovering cyanide worth 24 cts. from each ton of solution treated at a cost of 8.3 cts. per ton of solution, which is worth while.

b. Regeneration at the Loreto mill, Pachuca, Mex., according to R. R. Bryan and M. H. Kuryla in *Trans. 112, A.I.M.E.*, 1935, costs 22 centavos (6.2 cts. or 3.1 pence) per kilogram cyanide recovered, exclusive of credit for acid for treating filter leaves and for precious metals recovered. The plant treats 110,000 tons final wash solution a month. This carries 750 grams NaCN and 7 grams silver per ton; after treatment the tailings assay 70 grams cyanide and a trace of silver. Daily sulphur consumption is 2500 kilograms. Regeneration consists of (1) acidifying the solution with sulphur dioxide to neutralize lime and convert the cyanides to hydrogen cyanide; (2) vaporizing

the hydrogen cyanide from this solution by means of a large volume of air; (3) absorbing the hydrogen cyanide carried by this air in the regular mill solutions; (4) adding zinc dust to precipitate the gold; (5) recovering the gold-silver-copper precipitate by filtration.

At Rosario, Honduras.—According to P. T. Bruhl in *E. and M.J.*, Apr. 21, 1928, the Mills-Crowe cyanide-recovery process was started at Rosario, Honduras, in mid-1926 and in the first year saved \$21,000 in cyanide. In 1933 the silver content of the ore averaged 29 oz. per ton. It occurs principally as stephanite. Most silver ores need a strong cyanide solution to ensure satisfactory extraction. The regeneration process enables an increase in the strength of solution, but this causes additional expense through increased physical and chemical losses. Longer agitation and finer grinding may be more profitable than an increase in the strength of solution. After a certain extraction has been obtained (91.5 per cent in 1933), further improvement comes only by additions of cyanide out of all proportion to the final economic results. This cyanide-recovery process may be employed profitably in plants treating high-grade silver ore and in those situated in out-of-the-way districts where cost and transportation of cyanide are important considerations. But Bruhl questions whether it can be used profitably in plants treating lean gold ores.

From Gold-copper Residue Treatment.—H. B. Wright stated in *E. and M.J.*, May 5, 1923, that regeneration of cyanide from cupriferous cyanide solutions is profitable, easy of application and made possible the treatment of a dump of refractory residue in New South Wales, Australia. This material contained about \$4 gold per ton and 0.1 per cent copper in the sand and 0.33 per cent in the slime. While treating more than 10,000 tons of this mixture, half of the cyanide was regenerated, and about 1 lb. per ton copper was recovered.

To neutralize acidity, 4 lb. lime was added to each ton of sand going to the leaching vats; 2 tons cyanide solution per ton sand was used during the 14 days' treatment. But as it is essential for regeneration to deprive solutions of alkalinity or free cyanide, the sump solution titrating 0.06 per cent NaCN and 0.04 per cent CaO was applied to the 70-ton sand vats, a half ton to each ton of sand, preceding the neutralization by lime. The solution

draining away was neutral to phenolphthalein and was pumped to the regenerating part of the plant for subsequent treatment.

It was found best, by repeated tests, to apply one case or 224 lb. cyanide all at once to a vat containing 70 tons sand. The next step was to pump regenerated solution of 0.14 per cent NaCN and 0.10 per cent CaO strength over the cyanide placed on top of the sand, which dissolved all of the new cyanide required and raised the strength to 0.40 per cent. This solution was circulated by pump for 8-hr. periods over two days. Then it drained to the zinc boxes, having 0.15 to 0.20 per cent free cyanide.

Precipitation on zinc was for the richer and stronger solutions only, and 80 per cent precipitation of the gold was considered good work under the conditions of treatment. Calcium sulphate mud was a nuisance in the boxes. The gold-bearing sludge was so refractory that it was sold to a smelter.

Precipitation of gold from the weaker solutions was accomplished by sulphuric acid. Copper is present in the solution as the double cyanide $Cu_2(CN)_2 \cdot 2NaCN$ and is precipitated in the paddle agitator mentioned in the next paragraph, by means of sulphuric acid. The dried precipitate assayed 60 per cent copper, 70 oz. gold and double that of silver.

Regeneration was done as follows: The solutions neutralized and freed from active cyanide, as already mentioned, were eventually pumped to an elevated agitating tank of 50 tons capacity. Enough sulphuric acid was then added to throw down as cuprous cyanide, $Cu_2(CN)_2$, all of the copper present. Each charge of solution required from $7\frac{1}{2}$ to 14 lb. per ton. The agitator, from which an unpleasant odor arose, was then stopped, and the cuprous cyanide precipitate allowed to settle for $1\frac{1}{2}$ hr. The clear liquor, charged with hydrogen cyanide, was decanted into a 50-ton Dorr agitator. Milk of lime then was added until the solution showed 0.09 to 0.13 per cent free CaO and 0.14 per cent free NaCN. Agitation was effected by pumping. The regenerated solution was then pumped into the head tank and re-used in treatment, as described.

The Dorr agitator was cleaned of insoluble matter at periods of 3 to 6 months. The cuprous cyanide precipitate from the paddle agitator was allowed to accumulate during six charges, then was washed out and drained on a filter consisting of wooden

slats, coconut matting and sacking and finally dried and shipped to the smelter.

NOTE.—Although no ill or serious results ever attended the use of this process at this plant, it should be used with caution because of the extreme danger of cyanide poisoning by hydrocyanic acid gas.

LEAD SALTS AND LEAD MINERALS IN CYANIDATION

"Lead acetate is useful in assaying cyanide solutions, testing roasted ores for sulphur, correcting working cyanide solutions and in aiding precipitation," to quote M. W. von Bernewitz in *M. and S.P.*, Nov. 15, 1913. When soluble sulphides are formed by decomposition of certain minerals in ores, or sulphides are present as an impurity in the cyanide salt used, the addition of lead acetate will convert them to a sulphide of lead, which is insoluble and relatively innocuous. Other salts of lead, as the nitrate and the oxide litharge, are satisfactorily used.

When the existence of soluble sulphides in an ore or in the cyanide is known, it is well to add lead salts to the ore or solution at the start of experimental work. As an aid to precipitation, when the zinc-lead couple is used, gold may be precipitated from very weak solutions—as low as 0.005 per cent free cyanide. Lead salts may be added at any point in a pulp circuit and in the precipitating system and even to an ore before it is milled. Lead nitrate is used on the Rand, and the consumption during 1933 amounted to 0.03 lb. per ton ore milled.

GRAPHITE IN CYANIDING ORES

The so-called "graphite" or carbonaceous mineral that exists in certain ores in California, Montana, Ontario, West Africa and Western Australia has been a source of considerable difficulty in cyanide plants for many years. It usually occurs in graphite schists adjacent to the ore bodies and with quite an irregular amount of carbon, increasing the difficulty in plant control.

Several different treatment methods have been developed to overcome the premature precipitation of gold by the carbon from cyanide solutions. In West Africa, a post treatment was given following usual cyanide practice by re-leaching with sodium sulphide solution which acted as a solvent for the gold precipitated by the carbon. Results, however, were not satisfactory.

The Silver-Dorfinan process in which the ore, crushed in water, was treated with small quantities of fuel oil, kerosene or a combination of both, prior to cyanidation, was successfully used at some plants. The oiled graphite lost, to a large extent, its precipitating power.

Roasting was a satisfactory solution, but its cost precluded its use on many low-grade ores.

The later development of flotation for gold ores has, in most cases, provided a satisfactory pretreatment and has been applied in several ways. Flotation removes most of the freed carbon with the concentrate which is then roasted prior to cyanidation or shipped to a smelter. McIntyre, using flotation with cyanidation of concentrate, refloat their cyanidation tailing and return this concentrate to head of the cyanide circuit when they encounter carbonaceous ore.

Graphite is present in the gold-silver ore of the Missouri McKee Gold Mining Company, Madison County, Montana, but it is rendered innocuous by the treatment devised by R. J. Woody, mill superintendent. The ore after being crushed in water by 10 stamps fitted with 30-mesh screens flows over copper plates. The pulp is treated with kerosene from a drip, then passes over Wilfley tables and into an Akins classifier which makes a sand product and a slime product for cyanidation. The graphite, collected by the kerosene, floats atop the Dorr thickener and is periodically skimmed off. Assays of the scum show it to be valueless.

E. B. Leaver and J. A. Woolf of the U. S. Bureau of Mines described a number of experiments on California carbonaceous ores in *T.P. 481*, 1924, and some of their further work on the same subject was published in *R.I. 2998*, *U. S. B. of M.*, 1930.

FINE IRON IN MILL PULP

Fine iron and chips of iron and steel from crushing and grinding become mixed with the pulp and affect the subsequent treatment. At the Wright-Hargreaves, Kirkland Lake, two Dings high-intensity pulleys, which receive the tube-mill discharge pulp after dilution, extract as much as 40 tons per month of fine iron and steel. The material is cleaned on a small Wilfley table to free it of pyrite, gold and siliceous material, to prepare it for melting into balls.

USE OF BROMO SALTS

Bromo salts are a mixture of 57 per cent sodium bromide, NaBr, and 43 per cent sodium bromate, NaBrO₃, in the form of light-gray, light-yellow or reddish-brown crystals or powder. They are shipped in cases holding 224 lb. The individual salts have a density of 3.0 and 3.3, are soluble in water and for export are packed in 100- to 112-lb. cases.

The use of bromo salts for treating a telluride concentrate in the Wright-Hargreaves plant at Kirkland Lake is described by J. T. Willey in *E. and M.J.*, July 7, 1928.

Two 12- by 10-ft. tanks were used as collectors. They had mechanical agitators and a capacity of 15 tons solids and 15 tons solution. When thoroughly mixed, samples were taken for specific gravity, moisture, alkalinity and cyanide content. If necessary, sulphuric acid was added to reduce alkalinity to 0.1 per cent, and cyanide was added to increase the strength to 1 lb. per ton.

Bromocyanide was next made by mixing bromo salts, cyanide and sulphuric acid. To make 1 lb. bromocyanide it requires 521 grains bromo salts in 1500 cc water, 207 grams potassium cyanide in 1500 cc water and 486 grams 66°Bé. acid in 4500 cc water. Twenty pounds bromocyanide was made for each charge of 15 tons concentrate. The procedure was as follows:

Three 40-gal barrels were used. Into the first, which was immediately above the agitating tank, was put the requisite quantity of sulphuric acid. For 20 lb. bromocyanide this barrel held twenty times the quantity of water and acid given in the formula. The other two barrels were immediately above the acid barrel. From the bottom of each of these barrels was a lead pipe leading to the acid barrel, which itself had a lead pipe leading to the agitator. Plugs served as valves. The acid barrel was covered to prevent escape of bromine fumes and also had a hand-worked paddle mixer. Into one of the upper barrels was placed twenty times the quantity of bromine salts and water given in the recipe, and in the other barrel an equivalent amount of cyanide and water. The salts were then dissolved. Next, the plugs in the upper barrels were pulled out simultaneously, letting the solutions into the acid barrel, the contents of which were stirred. The 20 lb. bromocyanide was then run into the

agitator. Stirring proceeded for 24 hr. when the pulp was sampled and assayed. If high in gold, more bromocyanide was run in, and the agitation repeated until extraction was complete. During July, 1927, 70 tons concentrate averaged \$76 per ton before treatment and \$1.50 after treatment, equivalent to 98 per cent extraction.

HYDROGEN-ION CONTROL IN ORE FLOTATION

pH control is applied mainly to pulps in which certain reagents known as regulators are used. These reagents are capable of changing the pH concentration of the circuit and thus influence other factors involved in flotation. Their ability to depress or float selected minerals is closely associated with the degree of alkalinity (or acidity) of the medium. Frequent pH tests should be made. The use of excess lime can be avoided by proper pH control, and it may be pointed out that the use of too much lime not only wastes this material but may have a harmful effect upon the operation. In particular, lime tends to inhibit the flotation of metallic gold. The positive influence of hydrogen-ion concentration likewise holds true in the use of regulators other than lime; moreover, the degree of active alkalinity or pH exerts a characteristic influence throughout the processing not only on the regulators but also on the other reagents used.

Hydrogen-ion determinations should be performed on the individual raw ores, because these substances vary widely in their natural reaction, and a record of such characteristics will be of value to the operating engineer. Furthermore, pH determinations are of value in the control of the water used in flotation. Differences in pH always will be found in the raw waters, and these changes often cause variations in the operating results of the flotation machine which cannot be explained otherwise.

All crude ores will be found to vary in their lime-consuming power, and the extent of this factor should be determined prior to treatment. Provision must be made for the addition of enough lime not only to satisfy the lime-consuming power of the ore but also to produce the proper hydrogen-ion concentration within the flotation medium. In determining the lime-consuming power of raw ores, the following procedure has been recommended by La Motte, Kenny and Reed in their book "pH and Its Practical Application."

Determining Lime Requirements of Ores.—Fifty-gram samples of the dry, ground ore are placed in 8-oz. bottles with 200 cc distilled water. To the first sample bottle no lime is added; to the second the equivalent of 0.5 lb. calcium oxide per ton solution is added; and to the third, fourth, fifth and other bottles 1.0, 1.5, 2.0, etc., lb. calcium oxide per ton solution is added. The bottles are stoppered and agitated for 1 hr. after which the pulp is filtered, and the pH value of the filtrate determined. An analysis of the filtrate is then made to determine the amount of calcium oxide remaining in solution. When this value is known, the amount of lime consumed by the ore may be calculated. This procedure permits excellent control of the regulator, not only giving greatest economy but at the same time enhancing the efficacy of the other reagents used in the flotation.

Optimum pH Zones.—Lack of uniformity in untreated ores and differences in operating methods and in the types of separation desired make it impossible to prescribe optimum pH values for general use. Preliminary tests must be made on the ore. The nature of the separation involves considerable care in the selection of reagents as well as the proper alkalinity or pH of the circuit.

pH measurements may be made by the standard comparator method. Preliminary tests with satisfactory color charts will be adequate in outlining approximate value.

Plant Example 1.—At the Premier mill in British Columbia mine water is used in the plant that treats a gold-silver ore. It does not carry anything injurious to flotation and has a pH value of 7.2. Experiments showed that the recoveries were highest when a pH of 7.8 was maintained. Higher concentrations had neither detrimental nor beneficial results. As showing the value of pH controls, since 1929, the sodium carbonate reagent used was 0.4 lb. per ton ore in 1928, 0.08 lb. in 1929 and 0.03 lb. in 1931.

Plant Example 2.—The McIntyre-Porcupine takes its water from Pearl Lake. This water has a pH of 7.80 and total solids of 0.0473 per cent. Alkalinity determinations are made periodically with a Leeds and Northup indicator on the water and solutions in the flotation circuit. No compounds are used aside from the standard collecting and frothing reagents for the gold ore. The alkalinity of flotation solutions shown in the accompanying

table was given by J. J. Denny in *E. and M.J.*, November, 1933.

ALKALINITY OF FLOTATION SOLUTIONS AT McINTYRE-PORCUPINE

Place	pH
Tube-mill feed.....	8.10
Tube-mill discharge.....	8.90
Classifier overflow.....	8.70
Primary-cell feed.....	8.75
Primary-cell discharge.....	8.60
Secondary-cell feed.....	8.80
Secondary-cell discharge.....	8.40
Filtrate from concentrate.....	6.30

Plant Example 3.—Mine water is used in the Golden Chariot mill, Idaho. A pH of 8 is suitable for flotation; if it falls below this, soda ash is added to bring it up to this value.

SAMPLING ORE AND SOLUTION

Proper and accurate sampling is very essential to control of ore treatment. Until a plant has been running for some time much sampling should be done between the mine and the tailings discharge. Later some of these may be eliminated, or at least certain operations such as screen analyses in certain stages may be dispensed with.

Sampling may be divided into three different types according to whether broken ore, pulp or solution is being sampled. The most reliable sample is that obtained continuously. Various machines are made for cutting a continuous sample from an ore or pulp stream, whereas a drip-wire arrangement is usually used for solutions. If the sample cannot be taken continuously, some method of taking regular cuts is very often used. This may be done by machine or hand. Grab sampling is practiced but of course is not to be recommended for general use. For checking strengths of solutions and during test work grab sampling is often done and is useful. Some sampling practices at different plants are given in the following examples:

Golden Cycle.—Figure 81, from L. S. Harner's description of the Golden Cycle mill, *I.C. 6739, U. S. B. of M.*, 1933, shows how the ore is sampled prior to treatment. As this is a custom mill, the mill-head sample must be very representative, and much greater care is taken than is usual in a noncustom mill.

Hollinger.—The list on page 249 given by E. L. Longmore and M. E. Williams in *C.M.J.*, September, 1935, shows the routine samples taken in the Hollinger feed, the purpose and the

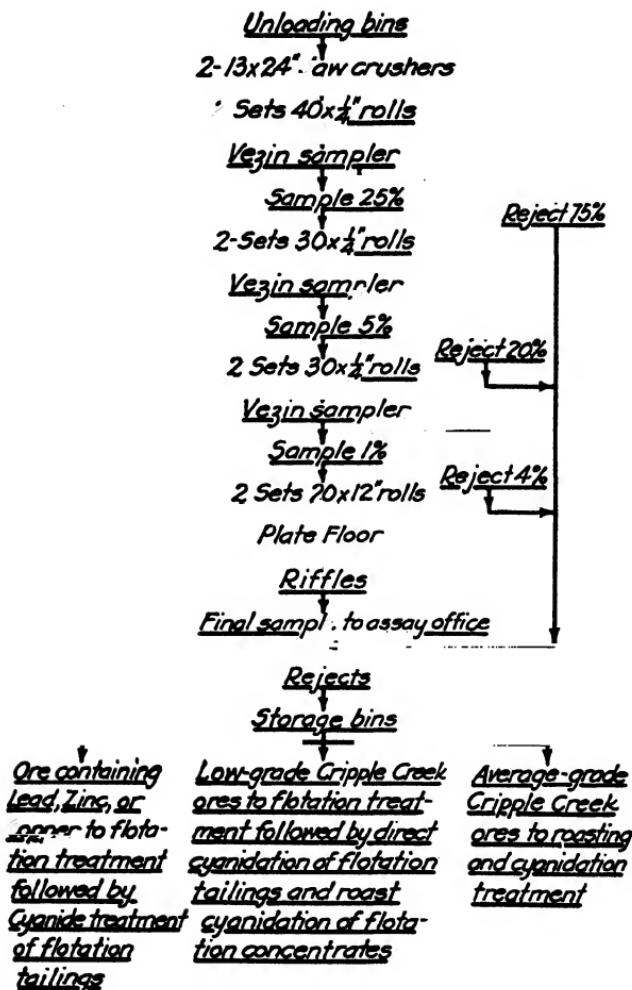


FIG. 81.—Golden Cycle sampling plant, Colorado Springs.

sampling interval. The Hollinger flow-sheet, given in Chap. XII, will aid in interpreting this list.

Longmore and Williams state that with the exception of the tailings from the secondary filters (final mill tailing), from which

an automatic sampler takes cuts at five minute intervals, all mill samples are taken by hand.

LIST OF ROUTINE SAMPLES AT HOLLINGER MILL

Product sampled	Purpose	Sampling interval	Assay-ton charge
Mill feed.....	Screen test	Hourly	1
Classifier Overflow; grinding circuit.....	Assay and screen test	4 times per shift at irregular intervals	5
Table head.....	Assay	2 hr.	2
Table concentrate.....	Assay	2 hr.	1
Table tail.....	Assay	2 hr.	5
Concentrate classifier Overflow	Screen test and assay	Hourly	5
Concentrate agitator head.....	Assay	2 hr.	5
Concentrate agitator tail.....	Assay and screen test	2 hr.	5
Dorr agitator head.....	Assay	2 hr.	5
Dorr agitator tail.....	Assay	2 hr.	5
C.C.D. tail (solution and pulp).....	Assay	2 hr.	10
Pachuca agitator head.....	Assay	2 hr.	10
Pachuca agitator tail.....	Assay	2 hr.	10
C.C.D. No. 3 tanks (last) thickener overflow grab sample.....	Assay	2 hr.	10
C.C.D. sump water grab.....	Assay	Once each shift	10
Gold solution.....	Assay	Every change-over of tanks	3 to 5 A.T. (Each shift)
Press tails.....	Color test	Hourly	750 cc
Press tails, continuous sample.....	Assay	Each shift	20
Repulped primary filter cake (pulp and solution).....	Assay	2 hr.	10
Primary filter cake solution.....	Assay	2 hr.	10
Secondary filter cake (pulp and solution).....	Assay and screen test	Automatic, 5 min.	20
Secondary filter cake.....	Per cent moisture	Hourly	
Secondary filter-cake filtrate (grab sample).....	Assay	24 hr.	10
Graphite taken from top of primary thickener 31 (grab sample).....	Assay		1
Press-tail drip (drip from gold presses).....	Assay	Once a week	10
Continuous sample.....			
Barren solution by-passes continuous sample during cutting in of gold presses.....	Assay	At end of cutin	10

About one ton of final pulp is collected in 24 hr. It is kept agitated in a small tank with an air jet. A suitable size portion is cut from this when discharged from the tank. Allowance is made in the assay for the small amount of gold dissolved during the 24-hr. collection.

McIntyre-Porcupine.—Sampling at this plant is described by J. J. Denny in *E. and M.J.*, November, 1933. Mill heads are sampled every half hour from the $\frac{3}{16}$ -in. product feeding the tube mills. Approximately 200 lb. is taken each shift and reduced to 50 lb. with a Jones riffle.

Flotation concentrate, flotation tailings and cyanide residue are sampled by Geary-Jennings automatic machines. About 100 lb. tailing and 50 lb. concentrate and cyanide residue are collected per shift. Hand samples are taken periodically by the operators from the classifier overflow and primary flotation tailings in the flotation section, together with classifiers, agitators, thickeners and filters in the cyanide section. Solutions in the cyanide circuit are sampled continuously with drip wires cutting through the flow. These samples are titrated for lime and cyanide and then assayed.

Head assays are usually about 1 to 2 per cent lower than actual bullion plus tails indicate.

Pioneer.—Sampling at Pioneer is described by R. Spry and P. Schutz in *Trans. 112, A.I.M.E.* The mill feed is sampled automatically by a Geary-Jennings sampler underneath the head pulley of the conveyor handling mill feed. The sampler cuts the ore stream every $2\frac{1}{2}$ min. and produces approximately 500 lb. per day. This is on a feed containing about 7 per cent $+\frac{7}{8}$ -in. lumps. This sample is crushed to $-\frac{3}{8}$ in. and riffled to 50 lb. before sending to the assay office for further reduction before assaying.

Recovery is based on heads calculated from bullion plus tails. The mill heads as determined from the sample usually are below the actual heads. Products into and out of all agitators, secondary thickeners and filters are sampled regularly and assayed daily to maintain close control of operations. This procedure is desirable because of the great variation in feed values at times.

Wright-Hargreaves.—The sampling methods at the Wright-Hargreaves, Kirkland Lake, Ontario, were described by Malcolm Black in *Bul. C.M. and M.*, September, 1935. Black notes the importance of accurate measurement of mill tonnage. The weightometer in the belt conveyor is checked periodically and is generally correct to within one-half of 1 per cent. Moisture samples are taken every hour. These are placed in a jar having

a tight-fitting cover so that drying does not take place before the moisture determination is made.

A unique system for sampling various pulps consists of several air-operated cutters, all controlled by one master controller. The master controller is made up of a timing mechanism driven by a Telechron motor, a Geco sampler, a four-way valve and an air header. The four-way valve is operated by the Geco sampler. The two pressure ports of the four-way valve are connected to a two-partition header. The cylinders operating the cutter are connected to the header by $\frac{1}{4}$ -in. copper tubing. Wedge-shaped cutters are operated by a cylinder and piston, being made to cut the pulp stream every 15 min.

Black also notes the need of a truly representative sample of press heads and tails. The common fault of a drip wire in the whole stream is that the rate of drip is not proportional to the rate of flow. The system at this mill is to use an automatic siphon in a small tank. This tank, holding about 1 cu. ft., is fed continuously by a $\frac{1}{2}$ -in. line from the main line. The tank discharges through a 1-in. pipe. The sample is taken from this discharge by inserting a $\frac{1}{4}$ -in. plug in the 1-in. line. The plug has a $\frac{1}{32}$ -in. hole for delivery of the sample into a large jar fitted with a rubber stopper. The sample size may be regulated by the distance the plug is inserted in the 1-in. pipe.

The sample bottles for press tails are kept in a safe so that if by carelessness the operator allows the press tails to run high, he cannot destroy the evidence. This safe consists of a metal box containing three Winchester bottles. The discharge pipe from the siphon passes through the box, and the sample is drawn off within the box. A small "drip catcher" operated from the outside is used to shift the flow to the various bottles.

General.—In taking samples of pulp containing cyanide solution, it is the practice at some plants to add a little potassium permanganate solution to the sample container to arrest the solvent action of the cyanide.

SIEVE OR GRADING TESTS

IMPORTANCE OF KNOWING PARTICLE SIZE

Wartenweiler, previously cited, made the following concise statement regarding grading tests:

This simple subject is important, particularly where extraction is dependent on degree of comminution and where large-scale milling technique must be largely governed by a measurement of particle size.

With regard to sieve tests, F. C. Bond and W. L. Maxson, in *Trans. 112, A.I.M.E.*, 1934, consider that more information is contained in a screen analysis than is ordinarily recognized and that need is increasing for methods of making this additional information easily available. As more comprehensive studies of crushing and grinding are made, and as the possibilities of decreased expense and increased recovery through a closer control of grinding are explored, methods of interpreting screen analyses assume an importance that they have not had heretofore. The information generally desired includes the size distribution of the material passing the finest screen, the presence of a hard-grinding fraction, the position of natural grain sizes, the presence of sundry materials with markedly different grinding characteristics, the presence and amount of finely divided material, as clay, which is merely unlocked in grinding and the total surface area of the ground product.

SIEVE SERIES

In the United States and in a number of other countries, testing sieves of the W. S. Tyler Company, Cleveland, Ohio, are the standard of metallurgists and technical institutions. This firm manufactures wire cloth to the standard scale. Many industries have established 200-mesh cloth as the minimum in screen sizing, and as the U. S. Bureau of Standards has standardized the 200-mesh sieve made from 0.0021-in. wire, having an opening of 0.0029 in. or 0.074 mm this sieve has been adopted as the basis of the Tyler standard screen scale. When it is necessary to carry an analysis finer than 200 mesh, as is done at Kirkland Lake, Ontario, for example, sieves can be procured as fine as 325 mesh. When discussing pulp sizes finer than 200 mesh, many millmen refer to them as so many microns. One micron is 0.001 mm or approximately 1/25,000 in. Material that remains on 200 mesh is about 74 microns in size; that through 200 and on 325 mesh is about 58 microns. The laboratory at Noranda, Quebec, has separated, by means of an elutriator, pulp down to 8-micron size. If material is 100 per cent through 200 mesh, it is roughly

equivalent to 85 per cent through 325 mesh. (See Chap. XV for comparison Tyler and I.M.M. screen scales.)

DETERMINATION OF GOLD IN CYANIDE SOLUTIONS

Measure 250 cc solution (8.571 assay-tons) or any other convenient volume which will contain a suitable amount of gold or silver. A few drops of strong NaCN solution may be added if it is desirable to strengthen the solution. Then add 1 to 2 grams zinc dust, and stir thoroughly. Add 30 to 40 cc of a 50 per cent solution of lead acetate and stir. Heat the solution until steam rises, but do not boil. Then add slowly about 25 cc of c.p. hydrochloric acid which is sufficient to cause a lead sponge to form on top of the liquid and to clear up the remaining solution. Boil gently for a minute or two without breaking the sponge, then gently form the sponge into a compact mass with a stirring rod. Decant the liquid, wash the sponge in cold water, squeeze dry with the fingers, wrap in lead foil, dry thoroughly on a hot plate and cupel in a muffle. If the ratio of silver to gold in solution is not high enough for parting, silver foil should be added with the lead foil.

Detection of Gold and Silver in Cyanide Solutions.—The following methods will be found useful for the rapid detection of gold or silver in barren or precipitated solutions. With practice, both estimations can be made with considerable accuracy but are, of course, intended to be supplementary to the regular daily assays.

GOLD (DOWSETT METHOD)

Take approximately 1000 cc solution in a tightly stoppered jar.

Add saturated solution of NaCN to bring the solution to be tested up to about 0.10 per cent NaCN.

Add 2 drops lead acetate (clear saturated solution).

Add a pinch (approximately 2 grams) zinc dust.

Shake well for about 2 min.

Pour into a large evaporating dish.

Settle and decant clear solution.

Add 10 cc aqua regia and evaporate nearly to dryness.

Take up in 2 cc concentrated hydrochloric acid.

Pour into a small test tube and cool thoroughly (*important*). The tube should be about 3 to 4 in. long and not over $\frac{3}{8}$ in. diameter.

Add a few drops fresh, saturated stannous chloride solution. The presence of gold will be indicated by a purplish ring at the point of contact or by a purplish tinge throughout if the tube is shaken.

The whole operation may be performed over a spirit lamp or similar flame and does not take more than 5 min. The presence of as little as 0.02 dwt. gold per ton solution is plainly shown.

Silver.—A few drops of a 10 per cent solution of sodium sulphide added to 25 or 50 cc of the plant solution to be tested gives a pure white precipitate of zinc sulphide in the absence of silver. The precipitate becomes brownish in the presence of silver and the depth of color is a close indication of the amount present.

The presence of the usual small amounts of lead in the solutions does not affect the result appreciably, according to the Merrill Company.

PYRITE AND PYRRHOTITE

The studies made by C. G. McLachlan and R. J. Morton at Noranda, Quebec, on the occurrence and behavior of pyrite and pyrrhotite, particularly in copper ore from the Horne mine, are of value to gold and silver metallurgists. Their findings are given in *Trans. 112, A.I.M.E.*, 1935. Later, experimentation on cyaniding the pyrite is given.

B. Robinson first conceived and demonstrated the possibility of the quantitative determination of pyrite and pyrrhotite in Noranda ore by chemical methods. The ore comprises 22 per cent pyrite, 52 per cent pyrrhotite and 7 per cent chalcopyrite. The methods of analysis are as follows:

METHOD OF PYRITE ANALYSIS

1. Weigh out into a 300-cc beaker a 1-gram sample of ground pulp, if the pyrite content is in excess of 10 per cent; or a 2-gram sample, if pyrite is less than 10 per cent.
2. Add 50 cc 2:1 HCl (2 parts water to 1 part HCl by volume).
3. Heat on hot plate for about 10 min., until evolution of H₂S fumes is finished as denoted by testing with lead acetate paper.
4. Filter with suction pump through a Gooch crucible lined with an asbestos pad.
5. Wash thoroughly with hot water.
6. Transfer crucible with contents to a 500-cc beaker, add a pinch of potassium chlorate to residue in crucible, then add enough concentrated nitric acid to decompose sulphides.
7. Evaporate to dryness on hot plate, and bake for 25 min.
8. Add 50 cc dilute HCl, and wash down with water.

9. Heat until nearly boiling, then add stannous chloride drop by drop until solution becomes colorless.
10. Cool solution and dilute to 300 cc bulk.
11. Add 10 cc concentrated mercuric chloride solution.
12. Add 15 cc titrating solution and 3 drops indicating solution.
13. Titrate with potassium dichromate solution to a permanent violet color.

NOTES.—(a) If considerable copper is present, the reduction of the HCl solution must be done by boiling with test lead. (b) Alternative to step 7: Evaporate until nearly dry. Add 10 cc concentrated sulphuric acid. Take to white fumes denoting removal of all nitric acid. (c) If alternative to step 7 is used, substitute the following for step 8. Cool and take up with 50 cc dilute hydrochloric acid. Wash down with water.

Solutions Required.

1. Standard potassium dichromate solution: 1 cc = 0.00466 gram Fe. 4.09 grams per liter. 1 cc = 1.0 per cent FeS_2 in 1 gram original sample.
2. Stannous chloride solution: Dissolve 150 grams stannous chloride in 250 cc concentrated hydrochloric acid, and then add 500 cc of water.
3. Mercuric chloride solution: Saturated solution mercuric chloride required (60 to 100 grams per liter).
4. Titrating solution: 150 cc sulphuric acid, 150 cc phosphoric acid and 700 cc water.
5. Indicating solution: 1 gram diphenylamine dissolved in 100 cc concentrated sulphuric acid.

For rapid work it is essential that a suction pump and a Gooch crucible be used for filtering. Also, it is advisable to use the alternatives to steps 7 and 8. The use of diphenylamine as an internal indicator was decided upon because of the necessity for speed, and its use has been entirely satisfactory. Application of the correction for copper in the determination of pyrite is as follows: A rapid determination of the copper in the sample is made by the fluoride-iodide method (H. N. Thomson, *E. and M.J.*, 1932, 133, 278). As in all samples dealt with the copper is present in the form of chalcopyrite, the percentage of iron in the chalcopyrite can be calculated from the chemical composition (Cu, 34.5 per cent; Fe, 30.5 per cent; S, 35.0 per cent) as follows:

$$\% \text{ Fe} = \frac{\% \text{ Cu} \times 305}{345}$$

The figure so obtained is then multiplied by the factor for pyrite, which is 2.146, and the product is deducted from the original

titration figure for a 1-gram sample; the remainder represents the percentage of pyrite in the sample.

Method of Pyrrhotite Analysis.—The method for pyrite analysis is followed up to and including step 4; the filtered solution from this step is then reduced by stannous chloride, and the iron content of the solution determined by following the same procedure as in the pyrite method. If the standard dichromate solution given above is used for determining the percentage of pyrite directly, it is necessary to multiply the titration reading by 0.750 to obtain the percentage of pyrrhotite as $\text{Fe}_{12}\text{S}_{13}$ in a 1-gram sample. (The formula for pyrrhotite given here naturally applies to Noranda ore and should not be used for another ore without being checked.)

METHOD OF MAGNETITE ANALYSIS

1. Weigh out into a 300-cc beaker a 1-gram sample.
2. Digest thoroughly with bromine and concentrated nitric acid on hot plate until no further reaction takes place.
3. Dilute with large quantities of water by successive decantations until all the iron in the solution is removed.
4. Add 50 cc 1:1 HCl. Warm until the magnetite goes into solution.
5. Reduce the solution with stannous chloride, and determine the iron content, following the same procedure as in the pyrite method.
6. Calculate the percentage of magnetite as Fe_3O_4 .

The application of the correction for magnetite to the method for the determination of pyrrhotite is as follows: The percentage of iron soluble in hydrochloric acid is determined by the pyrrhotite method. The percentage of iron in the form of magnetite is then determined on a duplicate sample by using the foregoing method. Subtraction of this figure from that obtained by the pyrrhotite method will give the corrected percentage of iron as pyrrhotite in the sample. This figure is now multiplied by the factor 1.622, and the result will be the percentage of pyrrhotite as $\text{Fe}_{12}\text{S}_{13}$.

CHAPTER XII

TREATMENT OF SIMPLE AND COMPLEX ORES

This chapter is devoted to brief descriptions of the current treatment of simple and of complex gold and silver ores. Many descriptions of plants and methods have been abstracted from recent issues of the technical press, but others have come to us direct. The examples given show the practice in different parts of the world in which the recovery of gold from gold ore may be made by the simple use of amalgamation alone or by the use of more elaborate combinations of two or more of the methods available such as amalgamation, gravity concentration, cyanidation and flotation.

The trend in metallurgical practice in such districts as Kolar, India; northern Ontario, Canada; the Mother Lode of California and the Rand of South Africa is reviewed. This and the description of individual plants in which the ores are amenable to ordinary methods of treatment are grouped under the heading "Treatment of Simple Ores."

On the other hand, gold ores that contain arsenopyrite, copper, pyrrhotite and the tellurides and the base-metal ores containing gold are refractory; therefore, the practice in recovering gold from such ores is described in the section on the "Treatment of Complex Ores."

Examples of the practice in treating silver ores in plants in North America are brought together in the section on silver. The refractoriness of manganese silver ores and the special methods required for treating them are discussed.

GOLD

SIMPLE ORES

Benguet Consolidated.—At Benguet, Philippine Islands, the Benguet Consolidated Mining Company (see Fig. 82) is milling 750 tons ore daily by the following process, as described by A. F. Duggelby, consulting engineer, in February, 1935, and as described in the paper presented at the February, 1936, meeting of the A.I.M.E. in New York by J. M. Morris, mill superintendent.

The ore consists of quartz, andesite, altered diorite, iron oxide and sulphide and clay, averaging about $\frac{1}{2}$ oz. gold and $\frac{1}{3}$ oz. silver per ton. Coarse crushing is done in a jaw crusher and a Symons cone crusher, with an Allis-Chalmers producing

$\frac{3}{4}$ -in. mill feed. Following the primary crusher, which is set at $2\frac{1}{2}$ in., is an Allis-Chalmers 6- by 10-ft. stone scrubber; 100 tons slime daily from the scrubber is floated in six Kraut and two locally made cells, using 0.05 lb. reagent 301 and 0.2 lb. pine oil plus sufficient sulphuric acid to maintain a pH of 6.6 to 6.8. Concentrates from flotation are sent to the secondary grinders, there joining the mill pulp for cyanidation.

First-stage fine grinding is done in two Allis-Chalmers 5- by 5-ft. ball mills and one Williamson 7- by 5-ft ball mill in closed

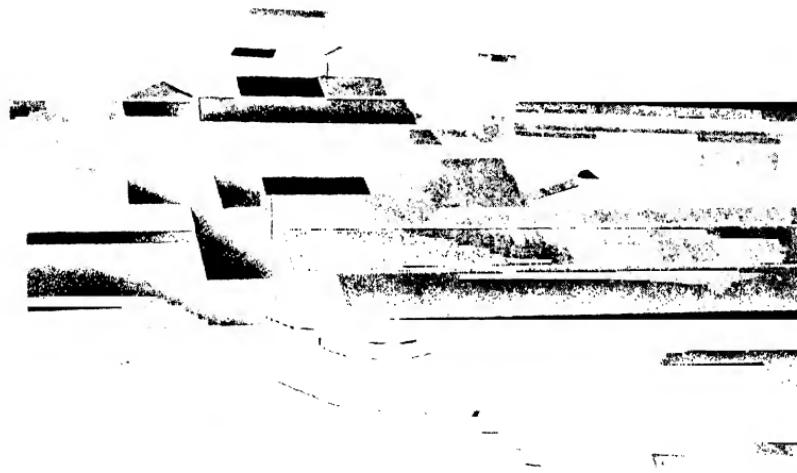


FIG. 82.—Benguet Consolidated, the oldest and largest gold property in the Philippine Islands.

circuit with two $4\frac{1}{2}$ -ft.-wide Dorr classifiers. Secondary-stage grinding is done in three mills, a 7- by 5-ft. Williamson, a 6- by 8-ft. Dewco and a 6- by 10-ft. Dewco Peb mill in closed circuit with a 6-ft.-wide Dorr FX classifier and a 16-ft. diameter Dorr bowl classifier. A screen analysis of the final classifier overflow shows 8 per cent plus 100 mesh, 20 per cent plus 150 mesh and 34 per cent plus 200 mesh cumulative.

Slime treatment consists of thickening to 60 per cent moisture in eleven 30- by 12-ft. Dorr thickeners, agitating in nine Dorr 30- and 20-ft. agitators for 48 hr., using 800 cu. ft. air per min. and filtering on four 12- by 18-ft. Oliver machines, cake of which carries 29 per cent moisture.

Consumption of cyanide is 0.95 lb., lime 7 lb. and zinc 0.105 lb. per ton ore treated. Cyanide is added to the first-stage ball mills. The loss of dissolved gold in the residue is 3.5 cts. per ton, and the loss of undissolved gold is 80 cts.; the loss of silver is 4 cts. Recovery approximates 90 per cent. The gold-zinc precipitate is melted without acid treatment, being fluxed with 28 per cent borax, 30 per cent silica, 22 per cent soda ash, 12 per cent hematite and 8 per cent manganese dioxide.

The main feature of this plant is the washing out of clayey material and floating the mineral from the slime, thus eliminating this troublesome constituent and considerably increasing the capacity of the plant.

A summary of the milling operation and benefits derived from the flotation of the primary slime may be stated as follows:

1. Total plant increased from 500 to 750 tons per day, with no increase in tank capacity in the cyanide unit.
2. Froth less than on the 500-ton basis with no slime losses through frothing.
3. Filter-cover life almost doubled.
4. Solution-pump life quadrupled.
5. Milling costs reduced 26 per cent over the previous 18 months.
6. The capital cost of increasing the plant from 500 to 750 tons per day with flotation was only slightly more than one-fourth of the cost of increasing the all-cyanide plant from 300 to 500 tons.

Big Jim (Oatman, Ariz.).—The countercurrent decantation plant of the Big Jim mine, flow-sheet of which is shown in Fig. 50, recovers 97 per cent of the gold in the ore, according to C. H. Johnson, in *I.C. 6824, U. S. B. of M.*, 1935.

The ore contains no sulphides and is hard and tough, consisting of, roughly, equal proportions of quartz and calcite, with small amounts of andesite wall rock. Gold is so finely disseminated in the quartz that unusually fine grinding is required to prepare it for cyanidation. The clay material in some of the ore, in addition to the tendency of the calcite to slime, makes the pulp difficult to settle, so that the settling area available is the limiting factor in the capacity of the mill.

Overflow from the Dorr classifier contains 15 per cent solids, 80 per cent of which is minus 200 mesh. The fine grinding is in cyanide solution, which accounts for about half the gold dissolution. The pulp is settled in a 24- by 8-ft. Dorr thickener,

and the discharge, containing 40 per cent solids, is delivered through a diaphragm pump to the first of a series of three Dorr agitators, one 19 by 14 ft. and two 13 by 14 ft. About 95 per cent of the gold is in solution by the time that the pulp leaves agitator 3. The treatment from this point is shown by the flow-sheet. Thickeners 2 to 6 are 15 ft. in diameter by 10 ft. deep, and their discharge is 50 per cent solids.

The precious-metal ratio of silver and gold is 1 to 2, and the bullion bars are 300 fine in silver and 640 fine in gold.

Chemical consumption per ton of ore is $\frac{3}{4}$ to 1 lb. sodium cyanide, 3 to 4 lb. lime, 1 oz. lead acetate and 0.16 lb. zinc dust. Water costs 30 cts. per 1000 gal. Direct labor cost is 80 cts. per ton, and total milling and cyaniding costs \$2.44, including 61 cts. for rental of the mill.

Carson Hill.—This mill, at Melones, Calif., was described by J. A. Burgess in *E. and M. J.*, March, 1935.

The ore consists of wide bodies of mineralized schist averaging \$2 or more. The treatment used is shown in Fig. 83a and b. The mill at present is handling 665 tons per day.

The ore is ground by thirty 1250-lb. stamps and three Hardinge pebble mills, two 8 ft. by 30 in., and one 8 ft. by 22 in. The stamps are fitted with $1\frac{1}{2}$ -in. screens and the pebble mills are run in open circuit. The largest particles discharged from the mill are said to be 0.1 in. A gravity concentrate is made, the tails sent to cyanidation and the concentrate regrind and amalgamated before going also to cyanidation. About 50 per cent of the gold is recovered by amalgamation in the stamp mill, and further 38 per cent is extracted in the cyanide plant. The average concentrate, before regrinding, assays \$30 to \$40 per ton. This is reduced to \$10 to \$20 before cyanidation.

Cyanide solution is carried at 0.4 lb. per ton. Milling costs are given as 56 cts. per ton, and over-all costs, including mining, milling, overhead, taxes and a proportionate charge for new machinery, are about \$2 per ton. Only the higher price of gold makes it possible to operate this property at a profit.

Day Dawn.—Near Wau, New Guinea, the Day Dawn (New Guinea), Ltd., has a 100-ton cyanidation plant treating a friable, gold- and silver-bearing quartz containing oxides of manganese. Two Empire ball mills can crush 65 tons per day. The flow-sheet shown in Fig. 84 is reproduced from *C.E. and M.R.*, June, 1934.

The Dentonia mine, Greenwood, British Columbia, according to W. Holdsworth in a paper presented in 1935 at the annual meeting of the B. C. Division of *C.I.M. and M.* and published

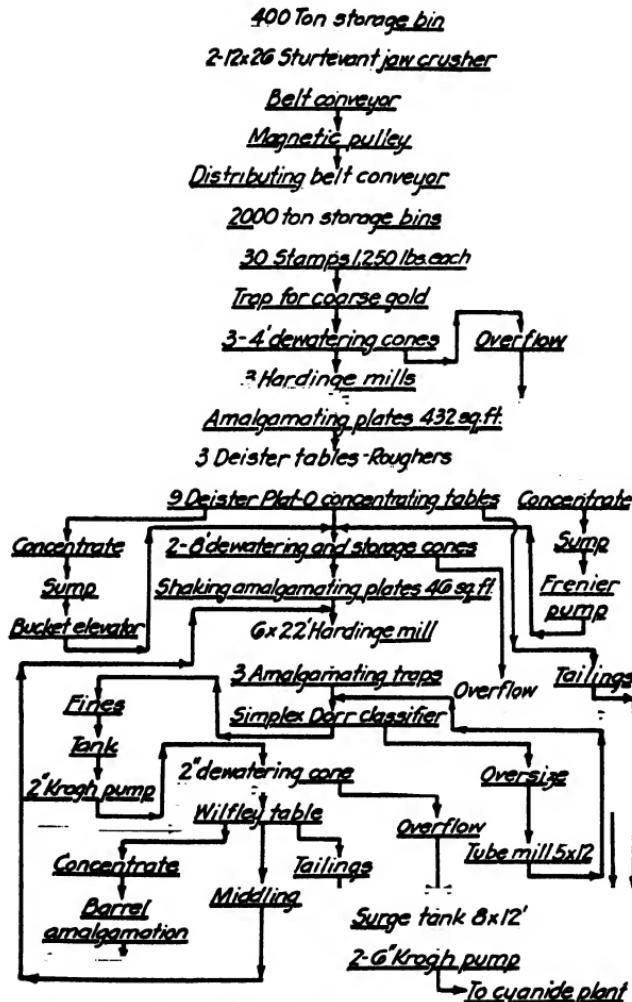


FIG. 83a.—Generalized flow-sheet of the Carson Hill mill.

in *The Miner*, Vancouver, November, 1934, yields an ore assaying 0.27 oz gold and 1.73 oz silver, with a little copper, lead and zinc. Some sorting is done before milling: The crushing plant, consisting of grizzlies and crushers, reduces the ore to $\frac{1}{2}$ -in. size. The

plant treats 110 tons a day. A 6-ft. by 48-in. Hardinge ball mill, in closed circuit with a 36-in. Akins classifier rotating at 5 r.p.m., produces 55 per cent through 200 mesh with a circulating load of 220 per cent. It is essential that the ore be not over-ground, for if it is too fine the losses increase. Aerofloat 31,

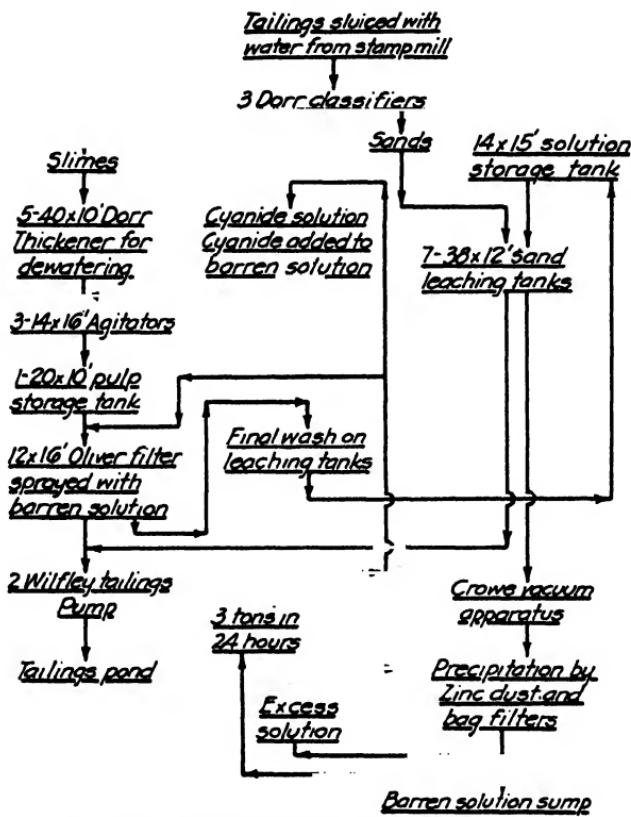


FIG. 83b.—Flow-sheet of cyanide plant, Carson Hill.

reagent 301 and soda ash are added to the ball mill, and the first two again at the Denver, Sub-A flotation machines each of which has six cells. Pine oil is fed at the head of the middling cells.

Oxidized ore distorts results from time to time, but ordinarily the recovery is 90 per cent and the ratio of concentration 30 to 1. Concentrates assay 65 oz. gold and 45 oz. silver. They are drained in 50-ton tanks having an ordinary filter bottom and are

then shipped by rail to a smelter. It costs \$1.35 per ton to treat the ore as described.

Golden Chariot Treatment.—After being idle for half a century, the Golden Chariot group of seven mines, near Silver City,

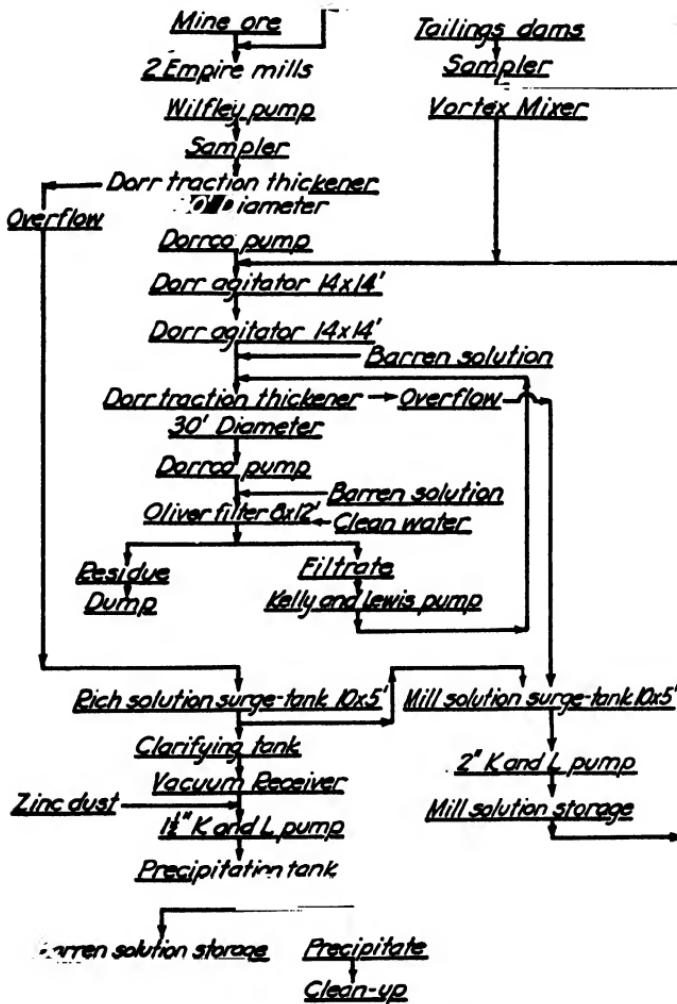


FIG. 84.—Flow-sheet of Day Dawn plant, New Guinea.

Idaho, has been provided with a 100-ton plant, as described by R. S. McClellan in *M. and M.*, November, 1934. The ore is quartz in granite, the gold is free and the silver appears as several

different sulphides. Freshly mined ore and old fills are being treated. Some hand sorting is done.

At present, the jaw crusher provides 1½-in. feed for the Hardinge 6-ft. by 16-in. ball mill, but a second breaker will reduce this to ½ in. Free gold is caught in a riffled launder and

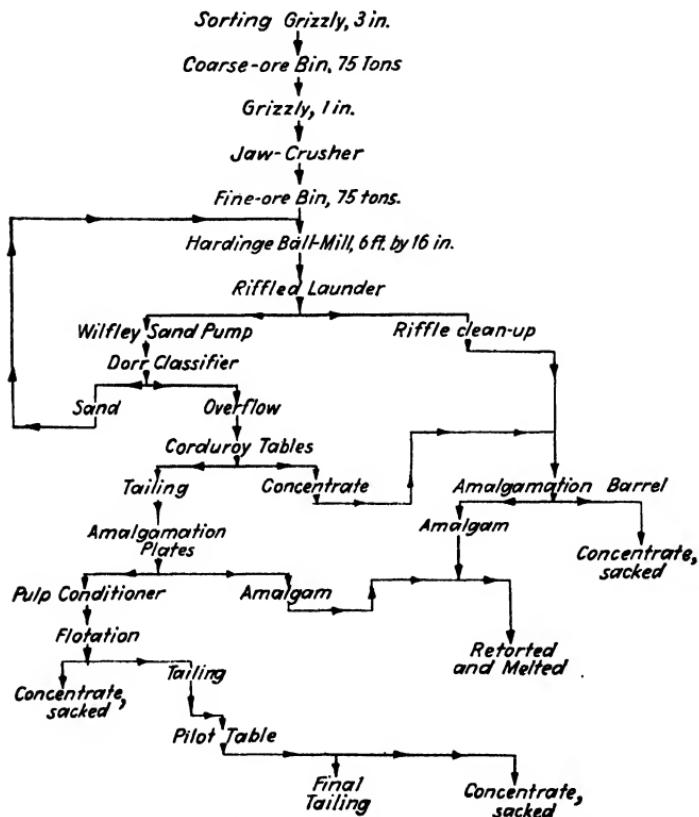


FIG. 85.—Flow-sheet of the Golden Chariot mill, Idaho.

on corduroy tables and copper plates. These recover 84 per cent of the total gold and 11 per cent of the silver. The ball-mill-classifier circuit produces 65 per cent of minus 200-mesh material. A Wilfley sand pump is part of this circuit, and a Krogh pump is used for lifting the final pulp into a conditioning tank for flotation. Aerofloat 15 (0.1 lb. per ton) and xanthate (0.05 lb.) are added to the launder which discharges into the Krogh

pump. Conditioning takes 8 min. Pine oil (0.09 lb.) is added to the pulp as it enters the second cell of a 6-cell Fahrenwald flotation machine. The pH of the water is maintained at 8. The concentrates, assaying 16 oz. gold and 638 oz. silver, are sent to a smelter in Utah. A Wilfley pilot table follows flotation and produces a concentrate assaying 0.4 oz. gold and 4 oz. silver. Flotation recovers 16 per cent of the total gold and 89 per cent

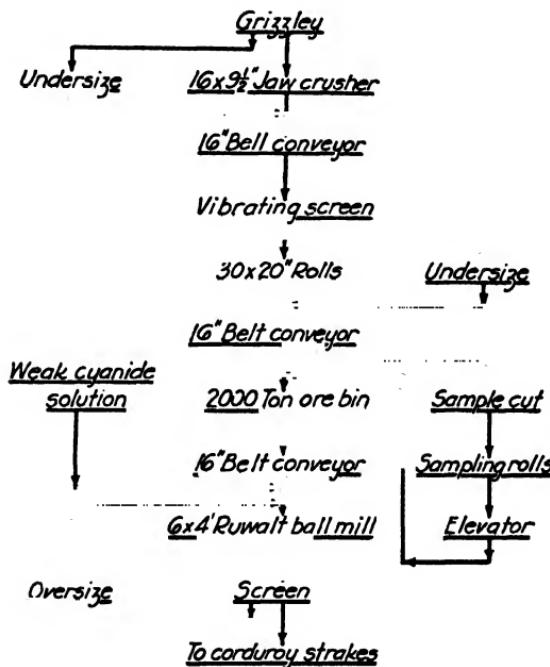


FIG. 86a.—Flow-sheet of Golden Plateau plant, Queensland. (First section.)

of the total silver. Milling cost is \$1.64 per ton. Figure 85 shows the flow-sheet of operations.

Golden Plateau.—At Cracow, Queensland, Australia, the Golden Plateau Company operates a 100-ton plant treating a quartz ore carrying fine gold. Figures 86 *a* and *b*, taken from *C.E. and M.R.*, July, 1933, shows the flow of the ore pulp. Features of interest are the rolls which reduce 2-in. crusher discharge to $\frac{1}{2}$ in., the use of belt conveyors to return the screen oversize to the ball mill and the two sets of corduroy tables, one following the ball mill, the other following the tube mill.

Golden Queen Mill.—The Golden Queen, a new mill in the Mojave region of the Southwest, is typical of the present trend in cyanidation.

Figure 88 shows the flow-sheet for treating about 325 to 250 tons per day. Feed (minus $\frac{1}{2}$ in.) goes to a No. 67 Marcy mill

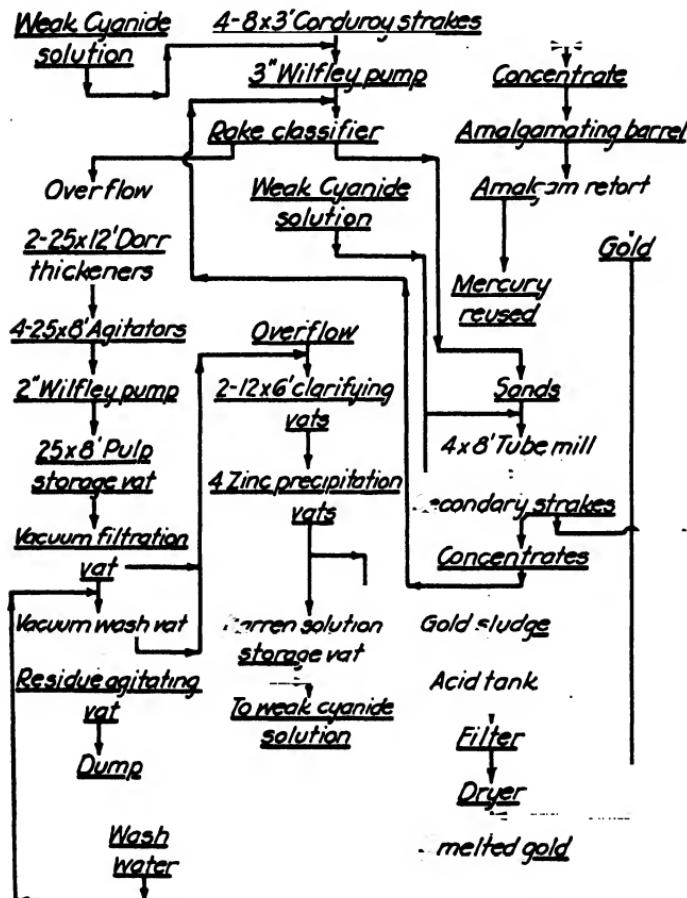


FIG. 86b.—Flow-sheet of Golden Plateau plant, Queensland. (Continued.)

in closed circuit with a 6-ft. Dorr duplex classifier. The overflow from this classifier goes to a similar grinding unit and the final grinding operation to 150 mesh is performed with a 67 Marcy mill in closed circuit with an 8- by 30- by 12-ft. diameter Dorr bowl classifier.

The primary 55-ft. Dorr thickener is followed by three 34-by 24-ft. Dorr agitators in series. Agitator discharge is washed

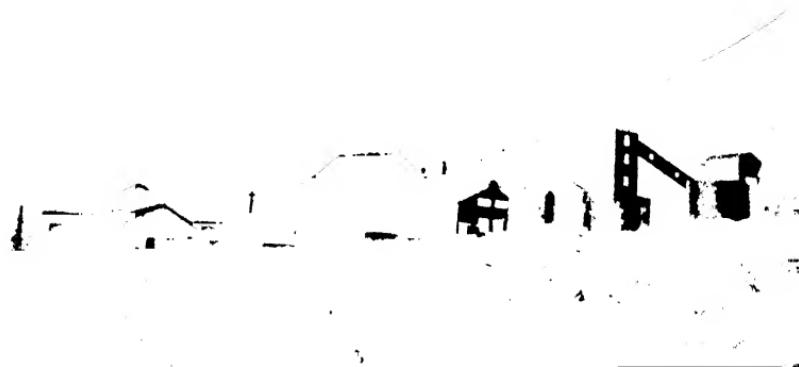


FIG. 87. Golden Queen mine, Mojave, Calif.

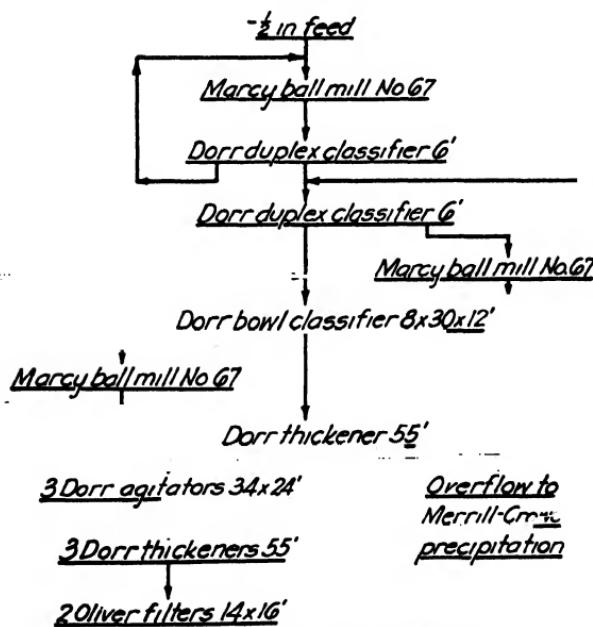


FIG. 88.—Flow-sheet, Golden Queen mine.

in three 55-ft. Dorr thickeners in countercurrent decantation and is filtered on two 14- by 16-ft. Oliver filters.

Merrill-Crowe precipitation is used.

Howey Treatment.—After the run-of-mine ore is sorted, and 18 per cent of it rejected, as described in the chapter covering sorting, the crushing plant of the Howey mine, Red Lake, Ontario, makes a product of which 39 per cent is plus 2 mesh and 38 per cent is minus 2 plus 6 mesh. The ore consists of quartz with 2½ per cent pyrite, with gold averaging nearly \$3 per ton. According to R. E. Sullivan in *C.M.J.*, October, 1934, liberation of the gold is effected by grinding to 40 per cent through 200 mesh in two 8-ft. by 48-in. and one 8-ft. by 6-in. Hardinge mill loaded with 19 tons and 20 tons of 5-in. balls. Crushing is done in cyanide solution, maintained at 1.1 lb. KCN and 0.55 lb. lime. The consumption of cyanide is about $\frac{1}{5}$ lb. per ton. Ore discharge from the two mills is pumped by Wilfley sand pumps into two duplex Dorr classifiers, and that from the third mill into a 54-in. Akins classifier. The rake product from all classifiers is split—10 per cent returning to the ball mills and the remainder to a 5- by 16-ft. Allis-Chalmers tube mill. The grinding load is 13 tons of 2-in. balls. Lime is added to the ball-mill return circuit, and consumption is nearly 1 lb. per ton.

All classifier overflow is divided into two treatment units, each made up of one Dorr tray thickener with diaphragm pump and five agitators in series. The thickener discharge has a dilution of 0.47 to 1. From the agitators the pulp passes to a surge tank, where it is kept in suspension by air and then filtered twice on American disc-type machines. Gold is precipitated on zinc dust using Merrill-Crowe equipment. Zinc consumption is 0.042 lb. per ton of ore. Treatment, excluding erushing, costs 54 cts. per ton.

Kolar, India.—Through John Taylor and Sons, managers, and a subcommittee of the chief reduction officers and chief cyanide chemists of the group of four plants operating at Kolar, state of Mysore, India, it is possible to give current treatment practice. (The detailed and excellent paper of R. H. Kendall and A. F. Hosking in *Bul. 242, I.M.M.*, 1924, or in *Proc. (Part 2) of the Empire Mining and Metallurgical Congress, London, 1925*, may be remembered by metallurgists.)

The Kolar group of mines is a remarkable one for depth attained (between 7000 and 8000 ft.), ore persistence, ore reserves, ore production (24,000,000), gold production (16,500,000 fine oz.) and dividends (£22,900,000 or \$110,000,000) to the

end of 1933. The ore of the four mines comes from the Champion lode or its branches and is free milling. The quartz is hard, and free gold is often seen. The average gold content in 1933 was 8.5 dwt. Galena, pyrrhotite and pyrite are accessory minerals but rarely exceed 1 per cent.

Using the Ooregum flow-sheet as approximating general practice we find the following: The coarse-crushing plant makes a $1\frac{1}{4}$ -in. product. This is done by a $1\frac{1}{4}$ -in. grizzly at the mine-shaft bin, a jaw crusher set at 2 in., another $1\frac{1}{4}$ -in. grizzly and a gyratory crusher set at $1\frac{1}{4}$ -in. All of the minus $1\frac{1}{4}$ -in. ore is trucked in ore cars and weighed before delivery to the mill ore bins. Between the crushers are two sorting belts from which 6 to 16 per cent of waste is removed. Some of the reject serves as tube-mill pebbles.

During 1933 an average of 188 head of stamps was crushing 31 of these at the Ooregum. Screens with nine holes per square inch (3 mesh) are general. Approximately, then, the $1\frac{1}{4}$ -in. feed is reduced to $\frac{1}{3}$ in. In place of copper plates in front of the mortar boxes, blanket tables are used to catch gold and sulphides.

The pulp is pumped by a 5-in. Allen centrifugal pump to a distributor, thence to two dewatering cones, the underflow of which goes to a sump that feeds a 10-in. Allen centrifugal pump. The latter pumps the thickened pulp, also old dump sand which has been pulped near by, to a tube-mill distributor. From this, the pulp flows to four primary and four secondary cones, their underflow feeding four tube mills. All of the ore at Ooregum and the other plants is slimed, the final product being 75 to 80 per cent through 200 mesh. (Formerly, two products—sand and slime—were made and treated separately.) In the tube-mill circuit are 20 blanket machines fed by a distributor; these save more gold and sulphides. All blanket concentrates are crushed and amalgamated, and the gold retorted in the customary manner.

Slime treatment consists of collecting and thickening in five cone-bottom tanks, pumping the thickened pulp first into four mechanical agitators for contact with cyanide and thence into a stock or surge tank. Filtration is a Butters-type filter installation the treatment cycle of which occupies 95 min. Residue is pulped and pumped to the dump.

Gold is precipitated from cyanide solutions on zinc shavings which are cut locally. Precipitation is practically complete.

Mill products are sampled at 14 points, and solutions at six points. The current ore averages 8.5 dwt. per ton, and the

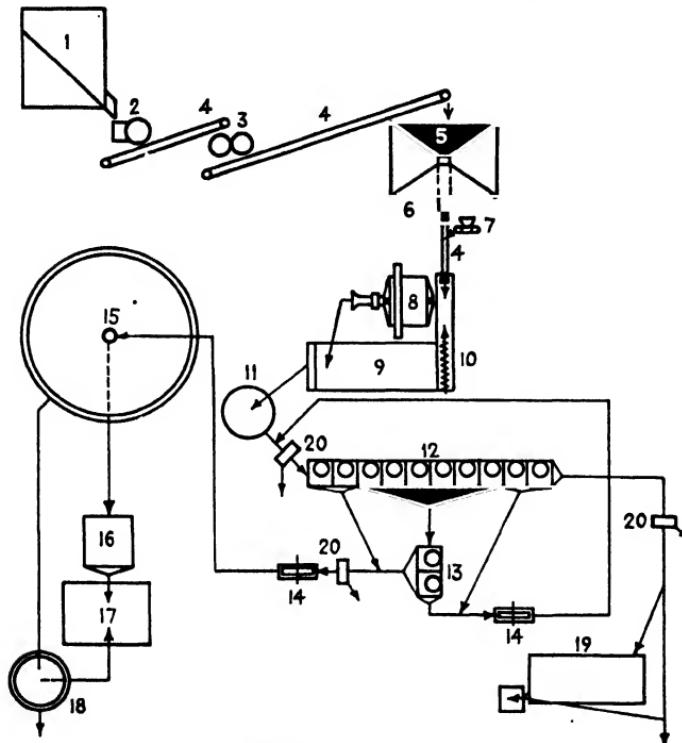


FIG. 89.—Flow-sheet of Lava Cap mill, California.

1. 100-ton coarse-ore bin in headframe.	11. 8' by 8-ft. conditioner.
2. 12'- by 20-in. Blake-type jaw crusher.	12. Kraut flotation roughers.
3. 42'- by 16-in. Worthington spring rolls.	13. Kraut flotation cleaners.
4. 16-in. conveyor belts.	14. 54'- by 8-in. Frenier pumps.
5. 200-ton fine-ore bin.	15. 18'- by 10-ft. Dorr thickeners.
6. 30-in. traveling belt-feeder.	16. 6'- by 6-ft. Oliver filter.
7. Dry reagent feeder.	17. Concentrates storage.
8. 8'- by 6-ft. Marcy ball mill.	18. 8'- by 10-ft. settling tank.
9. Model D Dorr classifier.	19. Deister Plat-O table.
10. 16-in. screw conveyor.	20. Automatic samplers.

residue 4.5 gr., equivalent to an extraction of 97.8 per cent, which has been maintained during the past 10 years. The cost is 52.5 pence or 105 ets. per ton. During 1933, 659,000 tons of ore and 499,000 tons of old tailings were treated in the four plants—Champion Reef, Mysore, Nundydroog and Ooregum. The consumption of cyanide is about $\frac{1}{2}$ lb. per ton ore.

Lava Cap Treatment.—The flow-sheet of the 300-ton Lava Cap mill, near Grass Valley, Calif., shown in Fig. 89, is from the article by J. W. Chandler in *E. and M.J.*, August, 1934. Two veins lie in slate, one being mainly gold bearing and the other mainly silver bearing, which is unusual in this district. The ore consists of slate containing quartz stringers, with free gold finely disseminated through the quartz and in pyrite, arsenopyrite and galena and silver as argentite associated with sphalerite.

The crushing and grinding equipment shown produces a pulp 73 per cent of which passes 100 mesh. A Denver unit flotation cell with cone classifier has been inserted in the ball-mill-classifier circuit. Copper sulphate, soda ash and sodium sulphide are added to the ball mill, the third reagent as a conditioner. Amyl and butyl xanthates and soluble starch are added to the rougher flotation machine, the starch to reduce excessive frothing. Pine oil is also added to the rougher machine.

The concentration ratio is slightly over 10 to 1, and the recovery in May, 1934, was 90.5 per cent of the metal value. Concentrate is trucked to the smelter.

London Mine.—Gold ore from the London mines, Alma, Park County, Colo., is treated as follows, according to H. L. Tedrow, general superintendent:

The ores come mainly from quartz veins between porphyry walls. Of late some ore has been mined from veins in limestone. This ore differs greatly from that of the quartz veins, as it contains more sulphides. Some of the veins contain varying amounts of graphitic shale, and all the ores contain the common sulphides pyrite, marcasite, galena, sphalerite and chalcopyrite. The gold is contained in the galena and pyrite and occasionally occurs free.

The ore is first washed on a 1-in. screen. The oversize, after sorting for waste, is crushed in a Blake-type crusher and a 4-ft. Symons cone crusher to $\frac{3}{4}$ in. The crushed ore joins the undersize of the 1-in. screen, is sampled, weighed on a Merrick weightometer and delivered to a storage bin. The crushing-plant output is 170 tons per day in one 8-hr. shift and comprises 30 tons sorted waste and 140 tons mill ore. The mill ore is sent by aerial tram to the mill storage bins. The undersize of the 1-in. washing screen is dewatered in a drag classifier before it joins the crushed

ore. The classifier overflow enters the mill feed by pipe line and goes to a Dorr thickener which likewise serves as a surge tank.

Grinding is done by a 6- by 6-ft. Marcy ball mill in closed circuit with a Dorr classifier. The overflow is 4 per cent plus 65 mesh and is the feed to the flotation section. The ball mill is equipped with a Weinig spiral screen with 10-mesh wire-screen cloth on the discharge end. The undersize of this screen goes to a Denver 500 Sub-A unit flotation cell which makes a finished concentrate and a tailing which goes to a No. 6 Wilfley table equipped with a Weinig wedge. A coarse pyrite concentrate containing much of the free gold is cut from this table and is dewatered and fed to the filter by a drag classifier. The tailing of the Wilfley table joins the oversize from the 10-mesh screen and goes to a 14-in. bucket elevator and thence to the Dorr classifier. Concentration at this point has as its purpose the quickest recovery of as much as possible of the free gold.

The overflow of the Dorr classifier enters the second cell of a 10-cell Hibbard flotation machine. A rougher concentrate is made which is pumped to a No. 1 cell for cleaning. The last three or four cells are usually run as a scavenger unit, and the concentrate made goes into a return circuit. The reagents vary considerably, but an average would be 0.67 lb. pine oil, 0.816 lb. Z-6 xanthate, 1.18 lb. fuel oil to depress the shale and occasionally 0.08 lb. sodium sulphide.

The tailing from the flotation section goes to a No. 6 Wilfley table. A coarse middling product is cut from this table and joins the concentrate from the scavenger cells to make a return circuit. This goes to a drag classifier, the overflow of which (all through 100 mesh) joins the flotation feed, and the dewatered sand discharge returns to the ball mill for regrinding. The flotation concentrate is thickened in a Dorr thickener and pumped to a 4- by 2-ft. Dorco filter where it joins the table concentrates before filtration.

Particular attention is paid to maintaining an even rate of feed to the ball mill and correct densities at the classifier and ball-mill discharge. The plant makes a recovery of 96 per cent of the gold.

Concentrates are trucked to the Arkansas Valley smelter at Leadville.

MOTHER LODE OF CALIFORNIA

The Mother Lode of California stretches along a distance of 100 miles of the foothill country of five counties. In general, the country rock consists of greenstone and slate. In places there is considerable carbonaceous schist. Gold occurs both coarse and fine in quartz veins and in the enclosing slates. In the quartz and slate is about 2 per cent of sulphides, mostly pyrite, with minor amounts of pyrrhotite, arsenopyrite, galena and others. At least two-thirds of the gold is amalgamable, the remainder being in the sulphides. For the California gold issue of *E. and M.J.*, November, 1934, J. B. Huttli reviews mining and ore treatment along the Mother Lode, and the tabulation shown on page 274 has been compiled therefrom.

This summary reveals a variety of processes for an ore of approximately similar composition throughout. Grinding to minus 80 to 100 mesh is fine enough for this ore.

ORE TREATMENT IN NORTHERN ONTARIO

Types of Ore Deposits.—The second edition, 1924, of *Ontario Gold Deposits, Their Character, Distribution, and Productiveness*, by P. E. Hopkins, of the Provincial Department of Mines, shows 24 lode gold areas, all in Pre-Cambrian rocks. To the end of 1934 these have produced 25,240,000 oz. gold. The pyrite-gold-quartz type of deposit, in which the pyrite is predominant, is the most common and most productive. It is represented by the Porcupine deposits. The gold-telluride veins, characterized by the presence of tellurides, as at Kirkland Lake, are another prominent type of deposit.

Kirkland Lake Ore and Its Treatment.—Ore from the mines of Kirkland Lake may be generalized as a silicified altered porphyry containing gold free and in pyrite, tellurides (principally altaite, a nonauriferous lead mineral), chalcopyrite and other important minerals. Some of the gold exists as a telluride, but Kirkland Lake ore as a whole is not a telluride ore, although some from the Lake Shore mine is considered as such. The tellurides are found throughout the ore zone and average less than 0.1 per cent. Pyrite amounts to 2 per cent or less. The ore is hard and requires fine grinding to liberate the gold. Gold extraction will average 95 per cent at a cost of around \$1 per ton.

Details of treatment will be found under other heads in this book.
The *Kirkland Lake Gold Area*, by E. W. Todd, Vol. 36, Part II,

SUMMARY OF ORE TREATMENT ON THE MOTHER LODE

Name	Place	Capacity, tons per day	Flow-sheet
Spanish.....	Washington	80 to 90	Jaw crusher; Hardinge ball mill and Dorr classifier; cyanidation in agitator, Hardinge thickeners, Oliver filters and Merrill-Crowe precipitation
Beebe.....	Georgetown	400	Hardinge-Hadsel gravity mills and Dorr classifiers to 80 mesh; flotation in Kraut machines; concentrates ground and cyanided; precipitation on zinc dust
Sliger.....	Georgetown	100	Stampers replaced by Symons cone-crusher; ball mill; flotation in Kraut machines; concentrates to smelter
Gold Reserve.....	Shingle Springs	125	Traylor gyratory crusher; Traylor ball mill and Dorr classifier to 100 mesh; cyanidation; precipitation in Merrill-Crowe bag-type unit
Montesuma.....	Plymouth-Apex	250	Marcy ball mills and Dorr classifiers with hydraulic traps in circuit; corduroy; Booth-Thompson flotation machines; tailings to waste, concentrate to smelter
Argonaut.....	Jackson	250	Jaw crusher; 60 stampers; amalgamation in boxes and plates; Johnson or Frue vanners; tube mills; shaking amalgamating plates; Deister and Wilfley tables; tailings to customs cyanidation, concentrates to customs plant
Boston.....	Jackson	150	Ball mill and Dorr classifier; modified Forrester flotation unit; Oliver filter
Kennedy.....	Jackson	150	Thirty stampers; amalgamation; tube mills; flotation
Carson Hill.....	Melones	700 (\$2 to \$3)	Jaw crushers; 30 stampers; amalgamation and hydraulic traps; Deister tables; concentrate over shaking amalgamating plates; pulp to Hardinge tube mills and Dorr classifiers, then over Wilfley table; tailings divided into sand and slime and cyanided, middlings ground in Hardinge tube mill, concentrates to amalgamating barrel; zinc-dust precipitation
Columbus.....	Tuolumne	35	Ten stampers; Deister table; Fagergren flotation machine
Pine Tree-Josephine	Bagby	100	Jaw crusher; Hardinge ball mill and classifier; Kraut flotation machine; 12-oz. concentrate to smelter

1928, of the Ontario Department of Mines, gives details of the ore deposits.

The accompanying typical analysis of ore from the Wright-Hargreaves mine is given by H. V. Wallace, Malcolm Black and Hector Farah in *Bul. C.I.M. and M.*, February, 1935.

ANALYSIS OF WRIGHT-HARGREAVES ORE	
Constituent	Per Cent
Cu.....	0.013
FeS ₂	1.780
Al ₂ O ₃	1.200
CaCO ₃	2.530
MgO.....	1.479
Te }	
Pb }	0.480
Mn }	
Mo }	
Insoluble.....	92.520
Au per ton at \$20.67 per oz	\$13 to \$14
Ag.....	Trace

Crushing.—Kirkland Lake coarse crushing is not so fine as the average at Timmins, Schumacher and South Porcupine. Cone crushers and screens are commonly used. At Teck-Hughes the mill feed is $\frac{3}{4}$ in. Rolls apparently are not popular, with Wright-Hargreaves the only user. In the Porcupine plants rolls make an even product.

Milling and Grinding.—All agree that Kirkland Lake ore requires fine grinding to liberate or expose the gold for cyanidation or flotation; hence, we note that practically every particle passes 200 mesh. Sieve sizes are discussed as minus 325 mesh or as microns. Finer grinding is contemplated. The ore is much harder than that in the Porcupine area.

All primary grinding is done in cyanide solution in Allis-Chalmers ball mills and in Marcy rod mills in closed circuit with Dorr classifiers, and secondary grinding in Allis-Chalmers tube mills in closed circuit with bowl and with rake classifiers. Ball consumption in the ball mills ranges from 1.9 to 5 lb. per ton, which is much heavier than at Timmins. Balls of cast iron and of forged steel from old rails are used.

Classification is by Dorr classifiers—single-stage and bowl types. Some of these carry a circulating load of 3 or 4 to 1, and the largest size is returning 7 to 1.

Sulphides.—The quantity and character of the sulphides vary in the same mine and in the different mines. Teck-Hughes has

SUMMARY OF ORE TREATMENT IN KIRKLAND LAKE DISTRICT

Operation	Lake Shore	Teck-Hughes	Wright-Hargreaves
Ore milled daily, tons.....	2300	1270	1000
Coarse crushing	Jaw crushers to 6 in.; cone crushers to 1½ and to ½ in., with screens	Jaw crushers, screens and cone crushers	Jaw crusher, screens, cone crusher, rolls
Sorting			
Size of feed to mills.	15 % on ½ in.	½ in.	2 % on ½ in.
Milling:			
1st stage.....	Ball mills using cyanide solution, in closed circuit with classifiers	Rod mills using cyanide solution, partly in closed circuit with classifier	Ball mills using cyanide solution in closed circuit with classifiers
2d stage.....	Tube mills in closed circuit with bowl classifiers	Tube mills taking part of rod-mill classifier rake product and in closed circuit with bowl classifier	Tube mills in closed circuit with bowl classifiers
Size of final product.	1 % on 200 mesh; 80 % through 325 mesh	1½ % on 200 mesh; 82 to 84 % through 325 mesh	99 per cent 200 mesh, 80 per cent 325 mesh
Sulphide handling..	Floated after cyanidation of bulk ore, ground and cyanided separately	None	Floated after cyanidation of bulk ore, ground and returned to main cyanide circuit
Thickening.....	Tray thickeners following 1st and 2d series of agitators	Two stages of thickening after agitation	Suction thickeners
Agitating.....	Three periods prior to first filtering, then again before second filtering	Two stages of agitation, followed by thickening	Agitators in series following thickeners, also single agitator between filters
Filtering.....	Primary filters, disc; secondary filters, drum	Two stages of filtering on disc machines	Two stages of drum filters with agitator between
Agitation hours before flotation....	19	23	21
Agitation hours of concentrate.....	85
Chemical consumption, pounds per ton:			
Cyanide.....	0.7	1.38	0.497
Lead salts.....	2.25 (litharge)		
Lime.....	3.5	4.49	2.814
Zinc.....	0.06	0.059 (os.)	0.096
Precipitation.....	Zinc dust	Zinc dust	Zinc dust
Recovery, per cent..	95+	94+	96+
Cost, cents per ton..	\$1+	96	\$1+

sulphides and tellurides yet employs straight cyanidation. Lake Shore floats after bulk cyanidation and cyanides the sulphide concentrate after regrinding; Wright-Hargreaves also floats after cyanidation but after special cyanidation returns the sulphide tailing to the bulk-cyanide circuit.

Slime Treatment.—Two stages of thickening and agitating (or agitating and thickening) and filtering are general. Lake Shore agitates three times. Compressed air aids agitation and aeration of the pulp. The pulp filters easily on American disc and Oliver drum machines.

Precipitation.—Clarification and deaeration of pregnant solutions and precipitation of the gold on zinc dust are general and satisfactory. The cleanup and melting follow standard practice.

Heating Solutions.—Solutions are heated to temperatures between 80 and 90°F. to increase extraction.

NOTE.—Milling practice has been undergoing rapid changes in this district in the last few years. Unless otherwise noted, the flow-sheets and data given in the following pages are as of about Jan. 1, 1935.

Lake Shore Mill.—The Lake Shore mill, the largest in the Kirkland Lake district, was originally an "all-sliming" straight-cyanidation plant but for a period running through 1934 added flotation to cyanidation tailing with separate additional cyanide treatment of the concentrate. As a result of experimental and research work beginning in 1933 and running through 1935, flotation and separate treatment of the sulphides were discontinued. At the beginning of 1936 straight cyanidation with improved chemical treatment was established and rendered combined cyanidation and flotation unnecessary. Finer grinding and increased time of agitations also have been proved desirable. According to A. L. Blomfield, managing director, flotation tests are being conducted with the sole object of floating sulphides enclosing gold which can be treated as far as now known only by roasting and recyanidation.

The results of the experimental and research work carried out at Lake Shore including detailed descriptions of methods used, test data and the new 1936 flow-sheet are fully described in a paper "Milling Investigations into the Ore as Occurring at the Lake Shore Mine" by the staff and published by the Canadian Institute of Mining and Metallurgy, 1936. The new flow-sheet

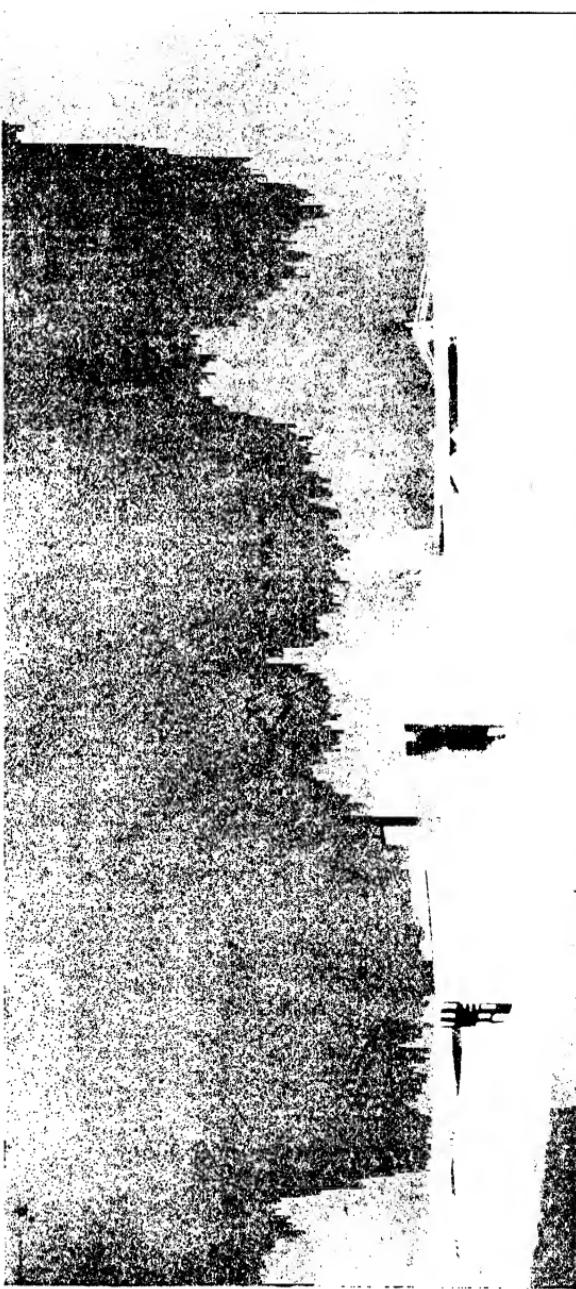


Fig. 90.—Lake Shore Mines, Ltd.; this famous property is at present the biggest gold-producer in the western hemisphere, with profits attaining \$8,000,000 per year.

is essentially the same when using the combined treatment as with the elimination of flotation and the concentrate retreatment section.

The following description, written in December, 1934, covers the combined treatment plant then operated at a capacity of about 2300 tons per day. (For flow-sheets see Figs. 91, 92, 93.)

Primary crushing is done underground by three 24- by 36-in. Traylor jaw breakers set at 6 in. Two $5\frac{1}{2}$ -ft. Symons gyratory crushers and two $5\frac{1}{2}$ -ft. Symons shorthead crushers set at $\frac{1}{4}$ in. complete the crushing. Niagara screens are used in a closed-circuit arrangement as shown in Fig. 91.

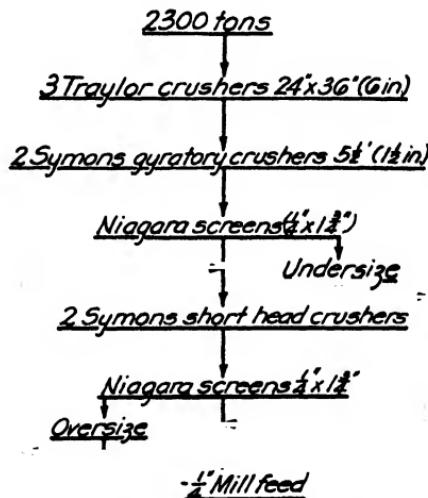


FIG. 91.—Flow-sheet, Lake Shore crushing plant.

Primary grinding in cyanide solution is carried out in seven 7- by 6-ft. Allis-Chalmers ball mills, using 4-in. balls and manganese liners in closed circuit with seven 4-ft. by 18-ft. 4-in. simplex Dorr classifiers, the overflow of which is approximately 20 mesh.

The primary classifier overflow joins the tube-mill discharge and is pumped to eleven bowl classifiers in closed circuit with eleven 5- by 16-ft. Allis-Chalmers tube mills. Five of these classifiers have 16-ft. diameter bowls with an 8-ft. rake width, and six have 15-ft. diameter bowls with a 6-ft. rake width. Manganese-steel balls $1\frac{1}{4}$ -in. in size are used in the tube mills.

Ball consumption for the ball-and-tube mills is about 4.5 lb. per ton feed.

Over-all grind is about 1 per cent plus 200 mesh and 80 per cent minus 325 mesh, and extraction during grinding is about 80 per cent.

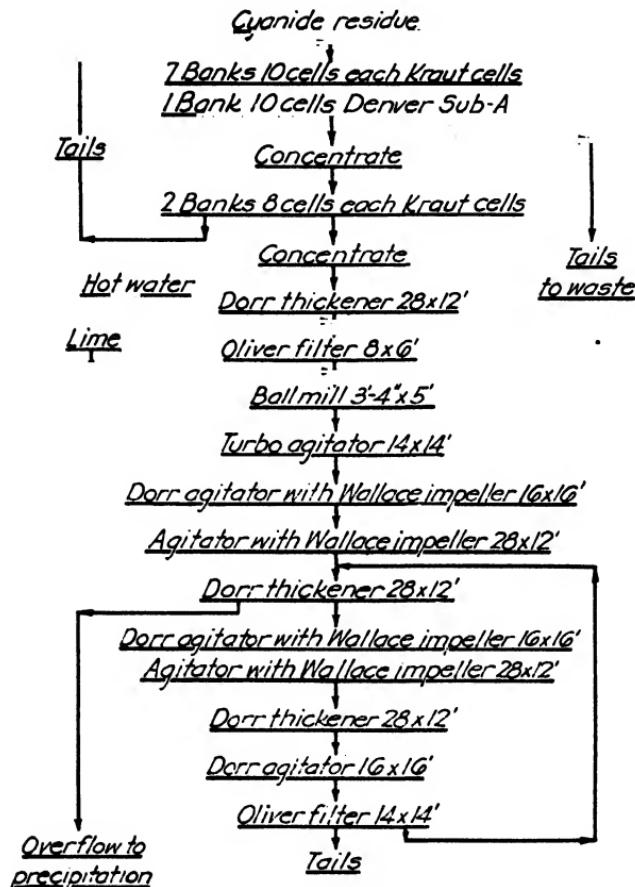


FIG. 92.—Lake Shore flotation plant flow-sheet.

The Dorr bowl-classifier overflow at about 10 per cent solids goes to two 30-ft. diameter by 24-ft. deep Dorr agitators, then to four 50-ft. diameter by 12-ft. deep Dorr balanced-tray thickeners. The overflow from these thickeners is precipitated, and the underflow is agitated with barren solution in seven 24- by 24-ft. Dorr agitators and three 30- by 24-ft. agitators. The pulp

is thickened in two 50- by 12-ft. Dorr tray thickeners and four 40- by 14-ft. Dorr tray thickeners. Overflow is used as grinding solution, and the underflow at 42 per cent solids goes to tertiary agitation in two 24- by 24-ft. Dorr thickeners. Primary filtration is carried out in eight 10- by 8-ft. American disc filters; the

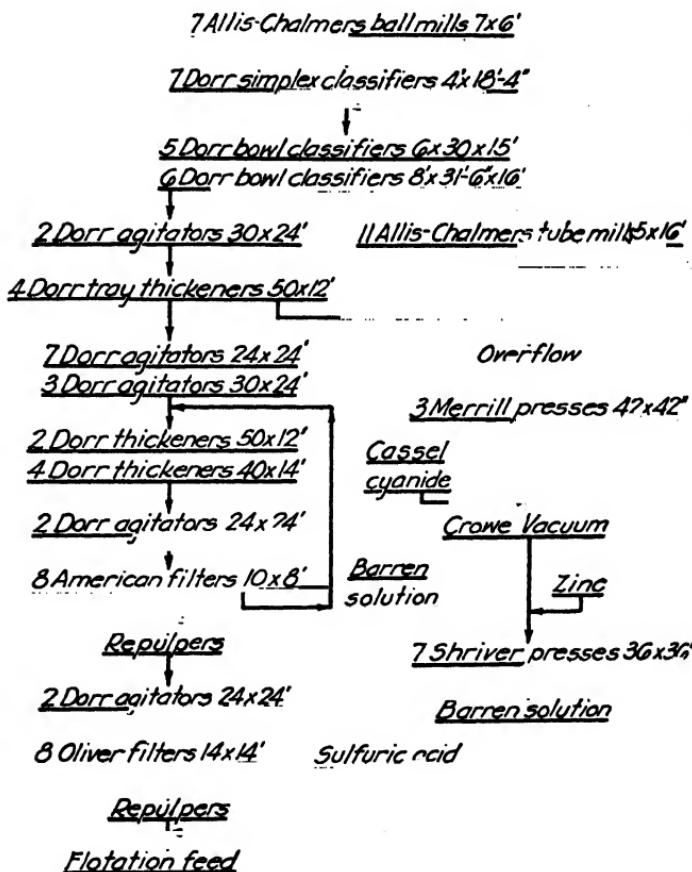


FIG. 93.—Lake Shore cyanidation circuit.

cake is repulped in barren solution, and two 24- by 24-ft. Dorr agitators serve as surge tanks before the secondary filtration; with eight 14- by 14-ft. Oliver filters. The cake is washed with barren solution and water.

The secondary filter cake is repulped, and sufficient concentrated sulphuric acid (3.8 lb. per ton) is added to give a pH of

about 8.5. This is the flotation feed. The acid is added to neutralize the depressing effect of combined lime and cyanide on gold-carrying sulphides. Kraut and Denver Sub-A flotation machines are used as shown in the flow-sheet. One series of 4 Fagergren cells are used.

Reagents used are No. 301, 0.18 lb. per ton; pine oil, 0.06 lb. per ton.

The cleaner concentrate is thickened in a 28- by 12-ft. Dorr thickener and then filtered on an 8- by 6-ft. Oliver filter with warm-water wash.

The concentrates and lime are mixed with cyanide solution in a 3-ft. 4-in. by 3-ft. ball mill in open circuit and agitated in a 14- by 14-ft. Turbo agitator, a 16- by 16-ft. Dorr agitator with an additional Wallace impeller and a 28- by 12-ft. agitator with Wallace impeller. The pulp is then given two stages of C.C.D. in two 28- by 12-ft. Dorr thickeners with two similar 16- by 16-ft. and 28- by 12-ft. agitators between. The final thickener is followed by another 16- by 16-ft. agitator and a 14- by 14-ft. Oliver filter.

Reagent consumption in the mill is reported as follows:

Cassel NaCN.....	0.06 lb. per ton ore
Aerocyanide.....	0.64 lb. per ton ore
CaO.....	3.5 lb. per ton ore
P60.....	½-lb. per ton to crusher product 2-lb. per ton concentrate
Zinc dust.....	0.0133 lb. per ton solution.

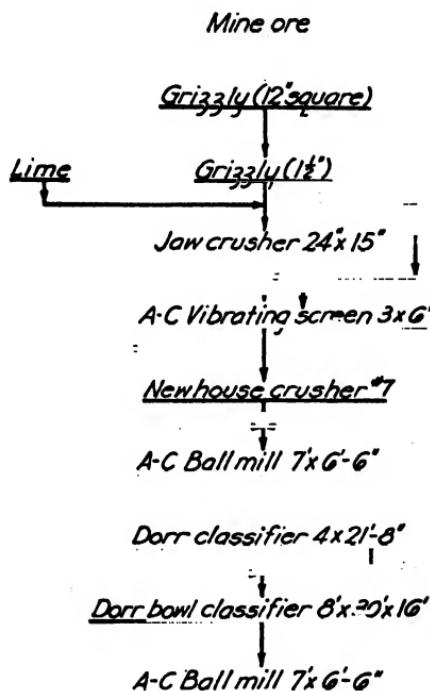
See chapter on costs for detailed milling costs with the new 1936 single-treatment flow-sheet.

Macassa Mill.—The mill of the Macassa Mines, Ltd., started operations on Oct. 13, 1933, and reached its rated capacity of 200 tons per day in April, 1934. The treatment process comprises bulk cyanidation, flotation of the cyanide residue and cyanidation of the flotation concentrate. The details of the Macassa operation, given herewith, are from the paper by H. B. Megill in *Bul. C.I.M. and M.*, September, 1935, and from information received direct.

A Ross feeder delivers mine ore to a sloping 1½-in. bar grizzly, the oversize passing to a 24 by 15-in. jaw crusher set at 1½-in. opening. The crusher product and the grizzly undersize are conveyed to a 3- by 6-ft. vibrating screen with ½- by 1¼-in.

apertures. The oversize, after passing through a No. 7 Newhouse crusher, joins the screen undersize and is conveyed to the 225-ton fine-ore bin in the mill.

The ore, reduced to $\frac{1}{2}$ in., is fed to a 7-ft. by 6-ft. 6-in. Allis-Chalmers ball mill operating at $24\frac{1}{2}$ r.p.m. in closed circuit with a



To thickeners

FIG. 94(a).—Flow-sheet, Macassa mill.

4- by $21\frac{3}{4}$ -ft. S. S. F. Dorr classifier set at a 3-in. slope and making 23 r.p.m.

The 28-mesh overflow from the primary classifier flows over a chip screen to a sump tank from which it is elevated by a 5-in. Allis-Chalmers rubber-lined pump to the secondary grinding circuit made up of a 7- by 6.5-ft. Allis-Chalmers ball mill, operating at $24\frac{1}{2}$ r.p.m. in closed circuit with an 8- by 30- by 16-ft. Dorr bowl classifier in which the pulp is reduced to 85 per cent through 325 mesh and only 1 or 2 per cent on 200 mesh.

The ball load is 25,400 lb. 4-in. cast balls in the primary and 24,800 lb. 1½-in. balls in the secondary circuit. The density in the primary mill is 77 per cent solids, and in the secondary mill 73 per cent solids.

Solution from mill storage is fed into the circuit at both ball mills and at each classifier. Solid sodium cyanide is added to

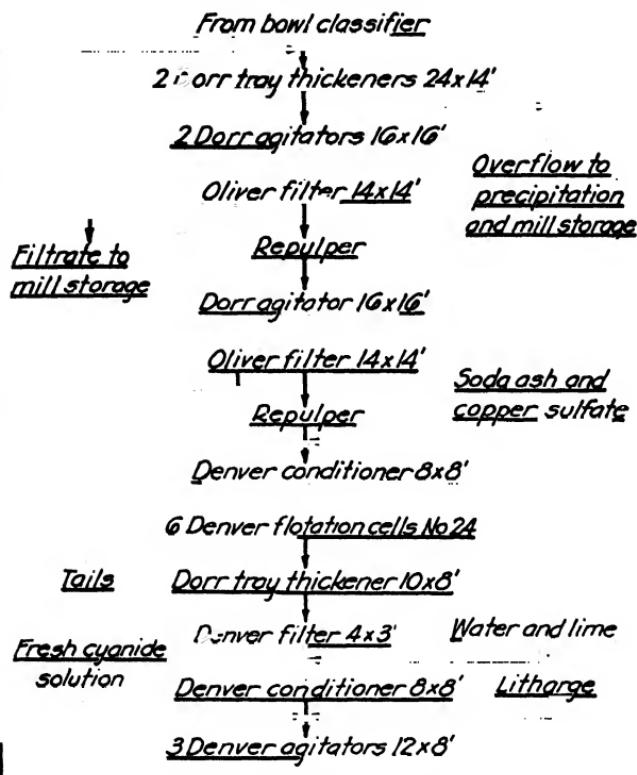


FIG. 94(b). Flow-sheet, Macassa mill.

the overflow from the primary classifier. (0.5 lb. NaCN per ton solution.)

The overflow from the bowl classifier passes over a chip screen and then is fed to two 24- by 14-ft. Dorr tray thickeners. The overflow from these thickeners goes partly to the precipitation circuit and partly to the mill solution sump. The overflow from the thickeners is delivered by two duplex Dorco pumps to two 16- by 16-ft. Dorr agitators in series.

After 22 hr. agitation the slime is filtered on a 14- by 14-ft. Oliver filter, the filtrate going to the mill solution storage, and the cake successively through a repulper and air lift to a 16- by 16-ft. Dorr agitator. The pulp from this agitator goes to a second Oliver filter, the filtrate being pumped with the other filtrate to mill solution storage. The cake from the second Oliver filter passes to a repulper in which are added soda ash and copper sulphate, the former to throw down all excess lime as CaCO_3 to avoid depressing the pyrite. The pulp at 2 to 1 density is delivered to an 8- by 8-ft. Denver conditioner in which potassium ethyl xanthate is added and is then advanced to a six-cell Fahrenwald flotation machine, the first four cells of which make 3 tons concentrate per 24 hr. The concentrate from the last two cells is returned to the feed end of the flotation unit. The flotation tailing is sampled by an automatic sampler and then passes outside to the tailing pond at a 2 to 1 density.

As a preliminary to cyaniding, the flotation concentrate is delivered to a 10- by 8-ft. Dorr tray thickener, the underflow from which is filtered on a 4- by 3-ft. Denver filter. The filter cake at 25 to 30 per cent is delivered to an 8- by 8-ft. Denver conditioner where water, slaked lime and sodium cyanide are added.

The overflow from the thickener and the filtrate go to the tailing pond. The conditioned pulp is pumped to three 12- by 8-ft. Denver agitators where fresh cyanide solution is added. The discharge from the third agitator is pumped to the 24- by 14-ft. Dorr thickeners where it joins the main cyanide circuit.

The Crowe method of zinc-dust precipitation is used.

The recovery is 95 per cent with milling costs of about \$1.45 per ton (1936). Mill heads run from \$8 to \$12 per ton. (Gold at \$20.67 per oz.)

The consumption of reagents is shown in the table on page 286.

During August, 1936, Macassa will complete the installation of an additional 26- by 12-ft. Dorr balanced-tray thickener and another 16-ft. Dorr agitator which will permit increasing plant capacity to about 250 tons with 30 hr. agitation. It is also planned to increase grinding to 90 per cent minus 325 mesh which may render flotation and separate concentrate treatment unnecessary.

Teck-Hughes Mill.—The Teck-Hughes is next to the largest plant in the Kirkland Lake district of northern Ontario. It

operates on a straight cyanidation basis in two sections, one of which, completed in 1931, has a capacity of 1000 tons a day, and the other, built prior to 1926, a capacity of 270 tons a day.

MILL SUPPLIES	Pounds per Ton Ore
Sodium cyanide:	
Agitation.....	0.411
Concentrate treatment.....	0.107
Lime:	
Agitation.....	2.152
Concentrate treatment.....	1.382
Zinc dust.....	0.106
Lead nitrate.....	0.005
Muriatic acid.....	0.180
4-in. balls.....	2.922
1½-in. balls.....	2.000
Soda ash.....	0.610
Copper sulphate.....	0.150
Ethyl xanthate.....	0.150
Alcohol, B23.....	0.030
B25.....	0.005
Litharge.....	0.010
Miscellaneous:	Gallons
Pine oil.....	0.001

The description of this mill and the accompanying flow-sheet (see Fig. 95) are from an article by Herbert N. Read in *C.M.J.*, December, 1931.

Average run-of-mine ore consists of altered and silicified syenites, lamprophyre and porphyry with 10 to 20 per cent quartz and a small amount of vein minerals, such as sulphides, carbonates and tellurides, the last being characteristic of the district. The average proportion of sulphides, of which pyrite is the most common, is about 2 per cent.

The ore is clean and practically devoid of clayey or colloidal matter. Gold occurs chiefly in the native form, associated with the sulphides and tellurides in minute particles. A small amount is present in the form of gold tellurides, calaverite and petzite. Four hundred-mesh grinding is required to free the gold, which is embedded in the quartz and silicified country rock. Mill heads average about \$15 (1931) per ton.

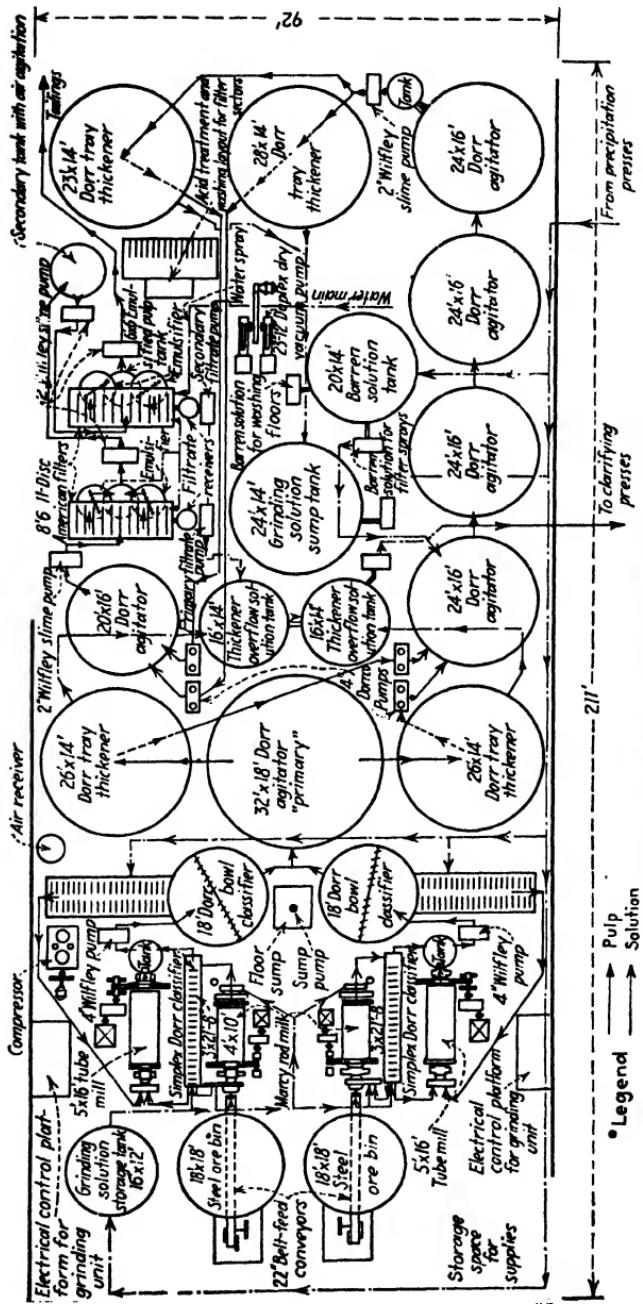


FIG. 95.—Flow-sheet, Teck-Hughes Gold Mines, Ltd.

Two crushing plants are used—the older one at the central shaft and the newer one at the south shaft.

At the central shaft, the ore is crushed underground by 24- by 15 in. jaw crushers to an 8-in. ring and then hoisted to the surface in 2-ton skips. Surface crushing is done in another 24- by 15-in. jaw crusher which delivers a 4-in. product to a No. 6 McCully crusher in circuit with Niagara vibrating screens, delivering a 1-in. product, via weightometer, to the mill bins.

At the south shaft, ore is hoisted, without previous crushing, to a steel bin on the surface, delivering 4-in. grizzly oversize to a pair of 24- by 15-in. jaw crushers, the first reducing to 8 and the second to 5 in. Crusher product and grizzly undersize pass to a $\frac{3}{4}$ -in. Niagara screen, the oversize from which is reduced to $\frac{3}{4}$ in. by a Symons cone crusher and then combined with screen undersize. The final $\frac{3}{4}$ -in. product then passes over a weightometer on its way to the mill bins.

The three units of the 1000-tons new mill are practically the same, and, as may be seen in the accompanying flow-sheet, each of the three mill units has two complete and identical grinding units. Therefore only one of the six grinding units will be described.

Crushed ore is reduced to a few per cent on 35 mesh by 4- by 10-ft. Marcy rod mills in partial closed circuit with 3-ft. by 21-ft. 8-in. Dorr classifiers. Classifier sands are split—40 per cent being returned to the rod mill and 60 per cent going to the tube mill with the classifier overflow.

Tube mills, 5- by 16 ft., in closed circuit with 6- by 30- by 18-ft. diameter Dorr bowl classifiers complete the grinding. Bowl overflow, at 4 to 1 dilution and only a per cent or two on 200 mesh, is pumped to agitation.

About 70 per cent of the gold is extracted in the grinding units. An appreciable concentration of values takes place in the bowl classifiers, sands assaying \$17.03 and overflow \$4.65 when milling \$15 heads. As is characteristic of Kirkland Lake ores, the bulk of the values lies in the minus 325-mesh product, this constituent having at times been \$44.45 per ton as against a \$12 a ton head and constituting over 40 per cent of the total value. (Gold at \$20.67 per oz.)

Bowl overflow receives about $5\frac{1}{2}$ hr. treatment at 4 to 1 dilution in a 32- by 10-ft. Dorr agitator. After thickening to about

59 per cent moisture in two 26- by 14-ft. Dorr tray thickeners, the pulp is pumped to four secondary agitators, 24 by 16 ft. arranged in tandem, and the overflow is sent to precipitation. Dilution during secondary agitation is 70 per cent moisture, due to the addition of barren solution to the thickener underflows. Between 17 and 18 hr. treatment is given.

Change of solution between primary and secondary agitation permits extraction to proceed to the end of the agitation system and yields final tailings with dissolved losses as low as 2 cts. per ton. Extra air is added during agitation, bringing the total air consumption to 1.8 cu. ft. per ton.

Pulp from the last of the secondary agitators is pumped to two 28- by 14-ft. Dorr tray thickeners from which the underflow is pumped to storage agitators, preceding an 8-ft. 6-in., 11-disc American filter. Primary filter cake is repulped with barren solution to 58 per cent moisture, pumped to an air agitated surge tank and thence to a secondary filter, duplicating the primary. Final filter cake is repulped with water and pumped to tailings disposal.

Barren-solution wash is used on the primary filter. On the secondary filter, barren solution is used on the first half and fresh water on the second half. Thickener overflow and secondary filtrate go to the grinding-solution return tank. Primary filtrate is pumped to the primary-thickener overflow-solution tank and thence to precipitation.

Pregnant solution from the primary thickeners and primary filters is pumped through Merrill central sluicing clarifying presses and thence to deaeration in a Crowe system. Zinc dust is fed by a standard Merrill feeder with an emulsifying cone. Precipitate is collected on standard Merrill presses. Precipitation is 3.4 tons solution per ton ore.

Wet precipitate is mixed with an unusually heavy flux—50 lb. borax, 45 lb. manganese dioxide and 25 lb. silica sand to each 100 lb. wet precipitate, containing 40 per cent moisture. Two hundred and fifty-pound charges of this fluxed precipitate are treated at a time in each of two Monarch "Rockwell"-type oil-fired furnaces.

Precipitate, on a dry basis, averages about 50 per cent bullion. Bullion, as shipped in bar form, has a fineness of 960, only 70 of which is silver.

Aero brand cyanide—49.5 per cent NaCN—is added dry to the tube-mill feed at the rate of 1.5 lb. per ton ore. Cyanide strength at the end of treatment is held at 1.2 b. KCN per ton solution.

Lime, containing 90 per cent available CaO, is added at the primary crushers at the rate of 5 lb. per ton. This is equivalent to a solution strength of 1.2 lb. CaO per ton.

Total extraction of gold, when treating \$15 heads, averages 94.07 per cent. The total duration of treatment is 31 hr.

Toburn Gold Mines.—The ore of the Toburn mine consists of porphyry, graywacke and quartz carrying about 0.57 oz. gold per ton. (Average heads, \$12 at \$20.67.)

The mill has an actual capacity of 350 tons per 24 hr. but has been operating on a 100-ton basis.

The mine ore is hand picked for steel and wood, and about 13 per cent of feed is rejected as waste on a 30-in. sorting belt which feeds a Buchanan 12- by 20-in. jaw crusher set at $2\frac{1}{2}$ in. The product of the crusher passes over a $\frac{3}{4}$ -in. grizzly the oversize of which is fed to a 2-ft. Symons cone crusher set at $\frac{5}{8}$ in.

The finished crusher product to mill bins is minus $\frac{1}{2}$ in.

The primary grinding is done in solution in a 6-ft. by 30-in. Hardinge ball mill, and the discharge passes to a 6-ft. by 25-ft. Dorr duplex classifier set at a slope of 2 in. per ft. and operating at 14 s.p.m. The rake product of the classifier is in closed circuit with a 5- by 20-ft. Allis-Chalmers tube mill. The overflow of the classifier passes to a 16- by 3-ft. Dorr hydroseparator, the underflow of which is returned to the grinding circuit and the overflow of which is thickened ahead of agitation. The hydroseparator overflow is all minus 200 mesh, 92 per cent minus 325 mesh and 37.5 per cent minus 10 microns.

The primary thickening is done in one 30- by 12-ft. Dorr thickener. Part of the overflow goes to precipitation, and part to the grinding circuit. The underflow at about 50 per cent moisture passes to three 16- by 12-ft. Dorr agitators in series where it is diluted to 60 per cent moisture with barren solution. The pulp is further diluted to 67 per cent moisture in the third agitator. It has a total of 24-hr. contact in the agitators.

From the third agitator the pulp passes to the first of four 28- by 12-ft. Dorr thickeners for C.C.D. washing. The last two decanting thickeners were recently converted to the balanced-tray type to increase settling capacity. Owing to lack of settling

VALUES AND HOURS OF TREATMENT AT DIFFERENT STATIONS

Location of sample	Value per ton	Percentage of extraction	Hours of treatment
Mill heads.....	\$15.00	0.00	0.0
Bowl-classifier overflow.....	4.50	70.00	1.0
Primary-agitator overflow.....	2.81	81.27	5.5
Primary-thickener discharge.....	2.63	82.47	2.0
No. 1 secondary-agitator overflow.....	1.81	87.93	4.3
No. 2 secondary-agitator overflow.....	1.39	90.74	4.4
No. 3 secondary-agitator overflow.....	1.16	92.27	4.3
No. 4 secondary-agitator overflow.....	1.08	92.80	4.4
Secondary-thickener discharge.....	0.88	94.13	2.0
Primary-filter feed..... cake.....	0.89 0.90	94.07 94.00	2.0 0.0
Secondary-filter feed..... cake (final tails).....	0.90 0.89	94.00 94.07	0.5 0.0
Totals.....	94.07	30.4

FINENESS OF GRINDING, TECK-HUGHES MILL

Tyler standard screens	Rod-mill feed	Rod-mill discharge, per cent	Simplex-classifier sands, per cent	Simplex-classifier overflow, per cent	Tube-mill discharge, per cent	Bowl-classifier sands, per cent	Bowl-classifier overflow, per cent
Plus 1.05 in.	2.79						
Plus .74 in.	12.67						
Plus .52 in.	18.63						
Plus .37 in.	19.78						
Plus 3 Mesh	10.87						
Plus 6 Mesh	13.21	0.14	0.25				
Plus 14 Mesh	9.08	8.43	10.53				
Plus 28 Mesh	3.53	22.04	30.48	0.34	0.03	0.54	
Plus 35 Mesh	1.20	12.29	16.39	1.59	0.12	0.96	
Plus 48 Mesh	1.06	9.97	12.38	3.34	0.33	2.73	
Plus 65 Mesh	0.48	3.72	3.69	1.07	0.23	1.76	
Plus 80 Mesh	0.58	6.28	2.11	5.87	2.17	6.17	
Plus 100 Mesh	0.65	4.97	3.08	6.25	4.08	9.40	
Plus 150 Mesh	0.67	5.17	3.54	9.14	11.50	24.72	
Plus 200 Mesh — 200 Mesh	0.43 4.28	2.71 24.28	1.54 10.41	6.55 64.95	13.30 68.24	24.30 29.42	1.68 98.32
Value.....	\$15.00	\$11.34	\$10.65	\$7.67	\$10.51	\$17.03	\$4.65

capacity in the first two decanting thickeners, normal wash solutions cannot be added, so some barren solution is added as the settling conditions permit.

The underflow from the final decantation thickener is again agitated in a 16- by 12-ft. Dorr agitator and filtered on one

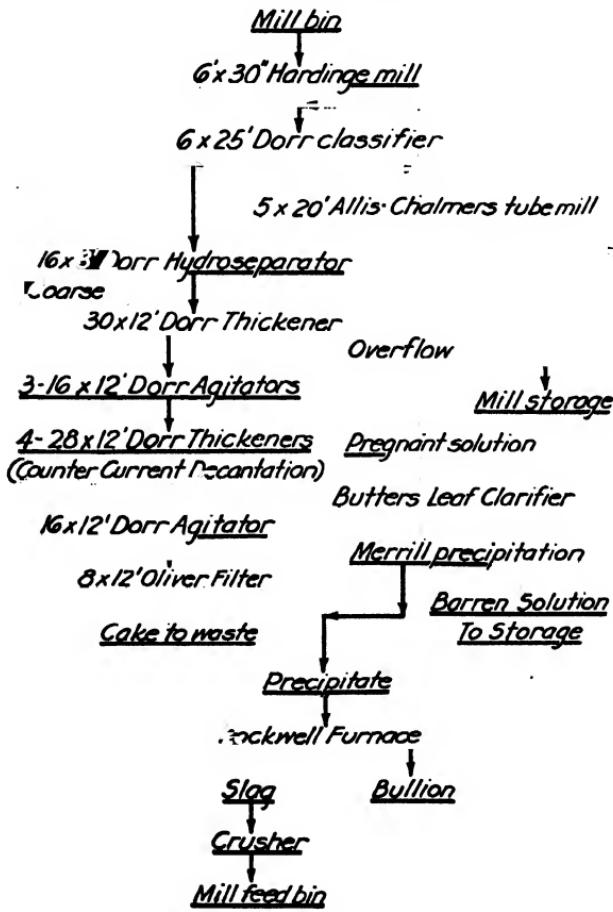


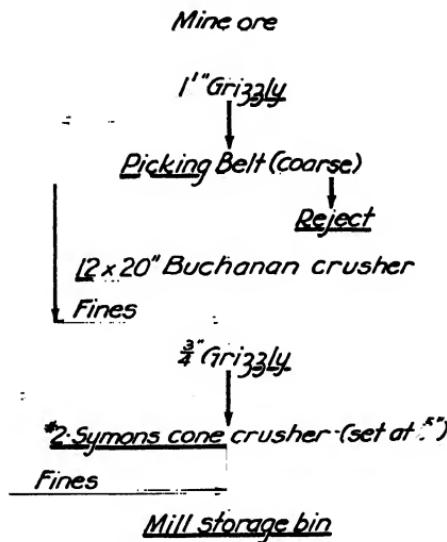
FIG. 96(a).—Cyanide plant, Toburn mill.

8- by 12-ft. Oliver filter where it is washed with 75 tons barren solution and 75 tons water before being discharged to waste. The moisture in the cake is 18 per cent. The dissolved loss in the tailing is about 1.5 cts., and the undissolved loss 40 cts. (Gold at \$20.67.)

Average extractions are as follows:

	Per Cent
Grinding circuit.....	84.
Primary thickener.....	3.0
4 agitators.....	5.6
4 thickeners.....	2.5
1 filter.....	1.5
 Total.....	 96.6

The solution is deaerated and precipitated by zinc dust. The pregnant solution carries from \$2.50 to \$3.50 per ton and is precipitated to a trace to 2 cts.; 360 tons of solution is precipitated



Approximately 10 tons barren solution per day is discarded per day to avoid fouling.

Wright-Hargreaves Mill.—In late 1933 mill alterations were completed, increasing the capacity of the plant from 800 to 1000

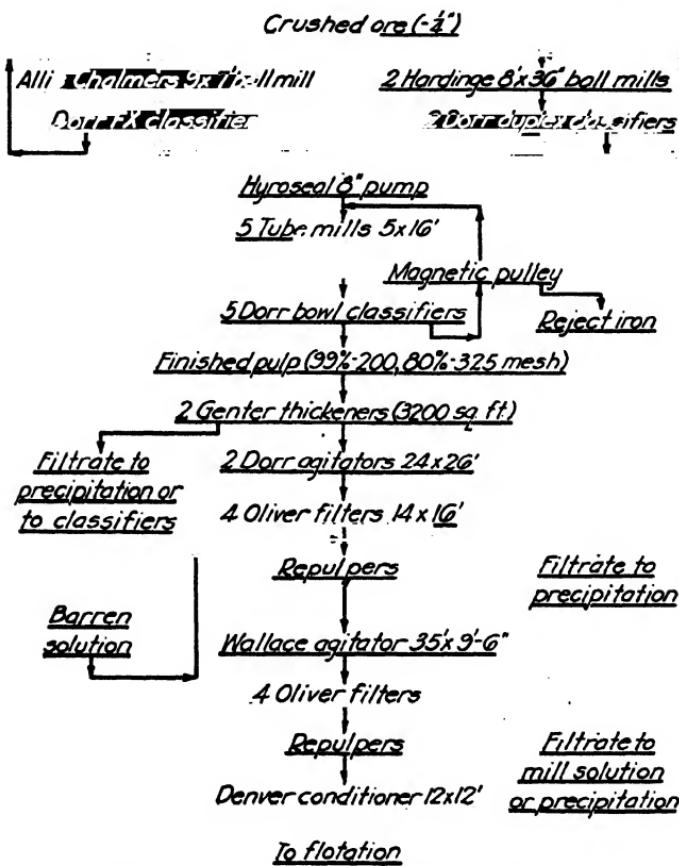


FIG. 97(a).—Wright-Hargreaves flow-sheet, cyanide section.

tons per day and changing the flow-sheet from straight cyanidation to cyanidation followed by flotation and retreatment of concentrate. As shown by the screen analyses in the chapter on grinding, the final pulp discharged from the grinding circuit is 99 per cent through 200 mesh and 80 per cent through 325 mesh.

The flow-sheet shown in Figs. 97a and 97b is reproduced from Donald F. Irvin's article in *Trans. 112, A.I.M.E.*, 1935, and other

data used here are from the same article, from the paper by the staff of Wright-Hargreaves in *Bul. C.I.M. and M.* for February, 1935, and directly from the management.

The overflow from the Dorr bowl classifiers, at about 10 per cent solids, flows by gravity to two Genter thickeners, one having 2300 sq. ft. and the other 2000 sq. ft. of filtering area.

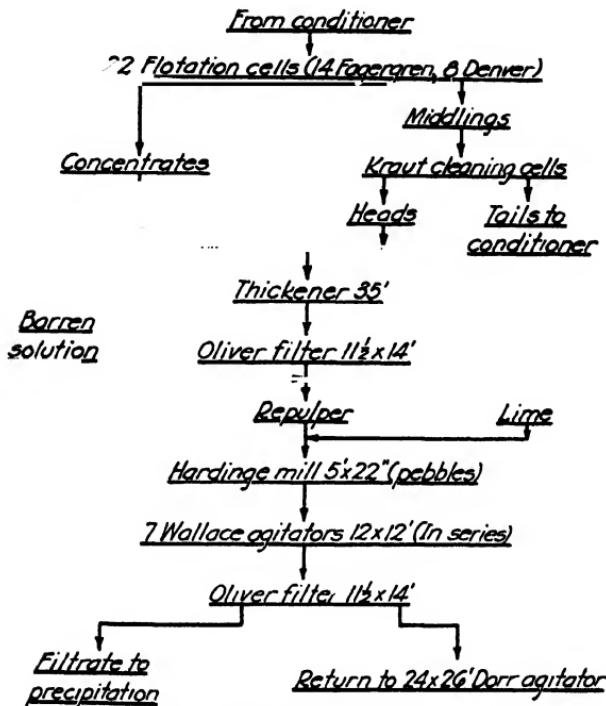


FIG. 97(b).—Wright-Hargreaves flow-sheet, flotation section.

The underflow, at about 55 per cent solids, is pumped from each machine by a Denver Quad diaphragm pump, discharging into the primary agitators (two Dorr machines 24- by 26-ft. and in series). The portion of the effluent from the Genter thickeners required for precipitation is pumped to Merrill sluicing clarifying presses, and the remainder sent to mill storage.

The overflow from the No. 2 primary agitator flows by gravity to four 14- by 16-ft. Oliver filters. A barren wash is used. The cake drops directly into a repulper of the Dye-Davis type,

repulping in barren solution. The pulp is next pumped at 50 per cent solids by means of a 4-in. Wilfley pump to a 33- by 10-ft. Dorr thickener which has been concerted to act as an agitator, and no further dilution is made. The pulp next passes to the second stage of filtration which, like the first, consists of four 14- by 16-ft. Oliver filters followed by repulpers. Water is used on the two front sprays, and barren solution on the others. The sprays on this bank are fitted so that all or any number may be

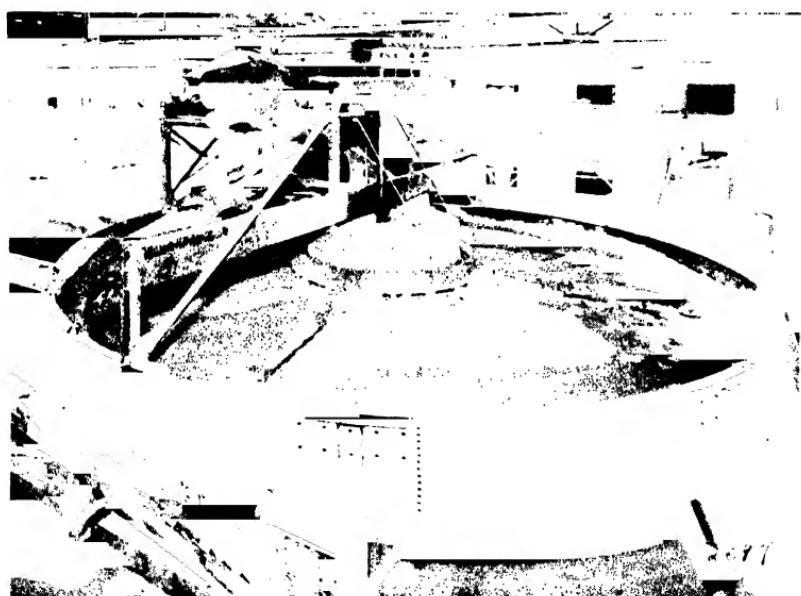


FIG. 98.—Large Dorr bowl classifier at Wright-Hargreaves, in closed circuit with secondary mills, overflowing at 98 to 99 per cent minus 200 mesh.

used for either water or solution. In order to prevent high dissolved loss, the filters must be scrubbed and acid treated about every second day. The scrubbing was formerly done by two men and required more than 2 hr. to make a good job of scrubbing and acid treating. A machine was developed on the property to do this work, and with its use one man can now do the job in about 30 min. Before scrubbing is started, the filter is reversed, and the scrubbing machine laid against the back at an angle of about 5 deg. As the scrubbing proceeds, weak hydrochloric acid is sprayed ahead of the brushes. The average life of the covers is a little more than two months.

FLOTATION.—After the second filtration the cake is repulped with fresh water, and copper sulphide and soda ash added at the repulper. One purpose of the copper sulphate is not so much to activate the pyrite but to precipitate any remaining dissolved gold.

It will be noted in the flow-sheet diagram that, following the last filtration, the pulp, after addition of copper sulphate and soda ash, is pumped to a 12-by-12 Denver conditioner. Before the pulp enters the latter it passes over a 4-by-4 screen of plate punched with $\frac{3}{16}$ -in. holes for the purpose of eliminating most of the remaining finely broken wood which may have tramped past crushers and classifiers. This wood used to give a great deal of trouble in the flotation circuit, as it formed a dense felt which gathered the rich concentrates, clogged impellers, blocked launders and was a detriment in various other ways.

The pulp entering the conditioner is diluted to 3 to 1, where Z-8 (a butyl xanthate) is added as a collector. The pulp is then fed to fourteen 56-in.-diameter Fagergren flotation cells in series but in two banks of seven cells each. B-23 and B-25 (duPont alcohols) with 20 per cent cresylic acid as a frother are added at various points along the series. The lips of the cells were cut down 4 in., and the gates of each of the cells simultaneously lowered. This has permitted the handling of the entire mill tonnage by these 14 cells in series, whereas, prior to the lip change, the pulp had to be divided equally between the two banks of 7 cells each. A finished concentrate is made from the first bank of seven Fagergren cells—such concentrate going to one 34-ft. 6-in. Dorr thickener. The pulp then goes to the second bank of seven cells, and the middlings from these go to a bank of eight Denver Sub-A cells for cleaning. The first four of the bank yield a finished concentrate. Middlings from the second four return to the top of the bank, and the tails go to waste. It was found that by adding a minute quantity of Minerec B to the top cell of the cleaners this reagent, with the residual duPont alcohols in the cleaner feed, gives a much better froth control. The finished concentrate joins that from the first bank of Fagergren cells at the 34-ft. 6-in. Dorr thickener.

The underflow from the thickener passes to either of two 11-ft. 6-in. by 14-ft. Oliver filters for dewatering prior to a partial regrind in a 5-ft. by 22-in. Hardinge silex-lined mill using

flint pebbles. Just before it enters the Hardinge mill, quicklime is added at the rate of 13 lb. per ton concentrates, and the discharge is pumped to a series of seven 12-by-12 Wallace super-agitators at 1.5 to 1 dilution, with a contact time averaging 86 hr. Of the precious metals, 85 to 90 per cent is extracted.

An interesting point is that, after this agitation in the retreatment circuit, the residues are returned to the primary agitators, and no building up of values takes place provided that not all of the sulphides is floated. It is noteworthy that in the flotation circuit not more than 0.5 per cent of the total sulphides contained in the flotation heads is floated, and the average pyrite content of concentrates is about 7 per cent FeS₂. For example, results over a month are as follows:

Flotation heads averaged.....	\$2.30 per ton and carried 1.78% FeS ₂
Flotation tails averaged.....	\$0.43 per ton and carried 1.36% FeS ₂
Flotation concentrates (by difference).....	\$1.87 per ton and carried 0.42% FeS ₂

If it is assumed that the values are all contained in the pyrite, this would work out as follows:

$$\begin{array}{l} 35.6 \text{ lb. at } \$2.30 = \$129.20 \text{ in heads} \\ 27.2 \text{ lb. at } \$0.43 = \quad 31.61 \text{ in tails} \\ 8.4 \text{ lb. at } \$1.87 = \quad 445.20 \text{ in concentrates} \end{array} \left. \right\} \text{per ton pyrite}$$

thus showing a depletion in grade or value contained in the pyrite through repeated action in one or other of the cyanide circuits. It should be borne in mind that there is approximately 18 to 20 cts. locked up in the tails that is not soluble in aqua regia.

The flotation heads average \$1.75, and the concentrates \$62. The dilution is 3 to 1. The reagents used in flotation per ton of ore averaged for a year as follows: Z-8 collector, 0.06 lb.; CuSO₄ conditioner, 0.25 lb.; soda ash, 0.09 lb.; cresylic acid, 0.02 lb.; and duPont alcohols, 23/25, 0.057 lb.

REFINING.—The average grade of precipitate is as follows:

	Per Cent
Gold.....	46.11
Silver.....	9.33
Copper.....	15.9
Zinc.....	9.31
Tellurium.....	6.29

Melting is done in two standard Rockwell furnaces, formerly using Carbofrax lining and fuel oil fired with air pressure at 90 lb. With tellurium and other base metals in the precipitate, it was found that a regulus was formed, due, it was believed, to the reduction of the oxides to the metallic state, effected probably by reaction with the carbon in the lining and the reducing action of the oil flame. Present practice is to use sillimanite lining which has entirely eliminated any regulus. Low-pressure air from a fan has been substituted for high-pressure air in the burner.

During 1935 an extraction of 96.8 per cent was obtained at a cost of \$1.279 per ton ore treated with a grind of 85 per cent minus 325 mesh.

Porcupine Ore and Its Treatment.—Ore from the mines of the Porcupine district may be generalized as quartz and mineralized schist, carrying very little to a considerable amount of free gold, and gold in pyrite which ranges from 2 to 8 per cent. There are other minerals, but they are of little significance, except pyrrhotite in the Dome. Treatment of these ores varies, as will be noted in the descriptions of representative plants. These plants report an extraction of 95 to 98 plus per cent at a cost of 62 cts. to \$1.20 per ton. There are other mills in the district, but those described represent the typical metallurgical practice. Volume 33, Part 2, 1924, of the Ontario Department of Mines, entitled "The Porcupine Gold Area," by A. G. Burrows, describes the ore deposits; and a section by G. E. Cole, E. L. Longmore, C. W. Dowsett and A. Dorfman explains mining and milling practice, although the latter has since been changed considerably.

Crushing.—Fine feed for the mills is made in the crushing plants, which makes it possible for a relatively small amount of fine-grinding equipment to handle large tonnages. Belt-fed rolls are a feature of each plant. They make a good mill feed at low cost. The cost of crushing ranges from 6 to 25 cts. per ton. At the Hollinger and McIntyre plants dust is drawn off from the screens and bins. At the Hollinger the cost for this operation is 0.37 cts. per ton. The dust collected is either dumped into the mill ore bin or is wetted and mixed with the ore taken to the mills.

Milling and Grinding.—At the Coniaurum and McIntyre plants the feed to the milling section is so fine that all grinding can be done in the tube mills in closed circuit with Dorr classifiers. Dome and Hollinger, presumably because the feed is coarser, use

ball mills and rod mills, respectively, preceding tube mills. Dome has a second stage of grinding in open circuit and a third stage in closed circuit on the cone underflow only. Hollinger pulp is finished in the second stage, with classifiers in closed circuit, but later, when the pyrite has been tabled, the concentrates are separately ground.

Free Gold.—Dome ore carries much free gold, some of it specimen ore. The latter is selected and treated separately. Corduroy tables catch 76 per cent of the gold. McIntyre ore also carries coarse gold, for the recovery of which a unit flotation cell and hydraulic cone are used in each tube-mill classifier circuit. The flotation unit is cleaned out once a day, and the concentrate treated. It scalps 40 per cent of the coarse concentrate.

Pyrite.—In effect, Dome, Hollinger and McIntyre practice is to recover the pyrite and then grind and cyanide it. At the McIntyre plant the mineral is floated, and all other pulp discarded. At the Hollinger the pulp is concentrated on tables, and the mineral saved for special treatment before it joins the regular mill pulp for further cyanidation. Dome procedure is to use cones to raise the sulphide content of the circuit and thus give the pyrite and pyrrhotite a selective grind and aeration with lime before cyaniding.

Slime Treatment.—Under this heading, which includes thickening, agitation and filtration, little comment is necessary. At the Hollinger both countercurrent decantation and double filtration are practiced. The Coniaurum mill uses double agitation and filtration, and the McIntyre employs triple agitation with filtering on American disc machines; the Dome uses Merrill presses which are gravity filled and the Hollinger has single and double filtering on Oliver machines.

Precipitation.—The gold from clarified and deaerated solutions is precipitated on zinc dust. An average vacuum of 26 in. is maintained in the Crowe tank.

Coniaurum Mill.—The milling methods of the Coniaurum mines, at Schumacher, Ontario, are described by John Redington in *I.C. 6541, U.S.B. of M.*, October, 1931, and by P. D. P. Hamilton in *Trans. 112, A.I.M.E.*, 1934, and form the basis for the following abstract and the accompanying flow-sheet (Fig. 99).

SUMMARY OF ORE TREATMENT IN PORCUPINE DISTRICT

Operation	Conisurum	Dome	Hollinger	McIntyre-Porcupine
Ore milled daily, tons.....	400	1500	Jaw crushers, screens, cone crushers, rolls $\frac{1}{2}$ in.	Jaw crusher, screens, roll 12 % on $\frac{1}{2}$ in. Minus $\frac{1}{16}$ mesh
Coarse crushing.....	Gyratory, screen, rolls			
Size of feed to mills.....	Minus 4 mesh			
Milling:				
1st stage.....	Tube mills using cyanide solution, in closed circuit with classifier		Rod mills using cyanide solution	Tube mills using water in closed circuit with classifiers
2d stage.....	None			None
Size of final product.....	70 % minus 200 mesh	83 % minus 200 mesh 78 % on corduroy	Tube mills in closed circuit with classifiers	57 % minus 200 mesh In unit rotation cells and hydraulic cones
Free-gold recovery.....	None	Cone underflow ground in tube mills in closed circuit with classifiers and then cyanided with the ore	Caught on tables and ground, then mixed with ground table tailings	Floated and cyanidized; re- mainder of pulp to waste
Sulphide handling.....	None	14- by 40-ft. tray thickener	40-ft. diameter tray thick- enera with 2000 tons treated by decantation	
Thickening.....	14- by 40-ft. tray thickener	8- by 40- and 14- by 40-ft. Pachuca agitators After corduroy and pulp conditioning	20- by 24-ft. Dorr and Pachuca type At mills	After floatation
Agitating.....	20- by 24- and 18- by 28-ft. agitators At mills	Double filtration on disc filters with agitation be- tween	3500 tons given double filtra- tion and 2000 tons single filtration rotary machines	Disc machines
Cyanide added.....				
Filtering.....				
Chemical consumption, pounds per ton:				
Cyanide.....	0.50	0.48	0.489	0.63
Lead salts.....	0.0075 (acetate)	0.03 (acetate)	0.0106	0.032 (acetate)
Lime.....	2.10	2.6	2.15	1.1
Zinc.....	0.021	0.045	0.05	0.084
Precipitation	Zinc dust 95 80	Zinc dust 98 \$1 + in 1933	Zinc dust 96 62	Zinc dust 96 70 in 1933
Recovery, per cent.....				
Cost, cents per ton.....				

The ore consists of a mixture of quartz and mineralized schist containing about 2.5 per cent sulphides.

The mill was designed to treat 500 tons ore per day but is now treating only 480 tons ore averaging 4.50 dwt. per ton in gold at

Mine ore

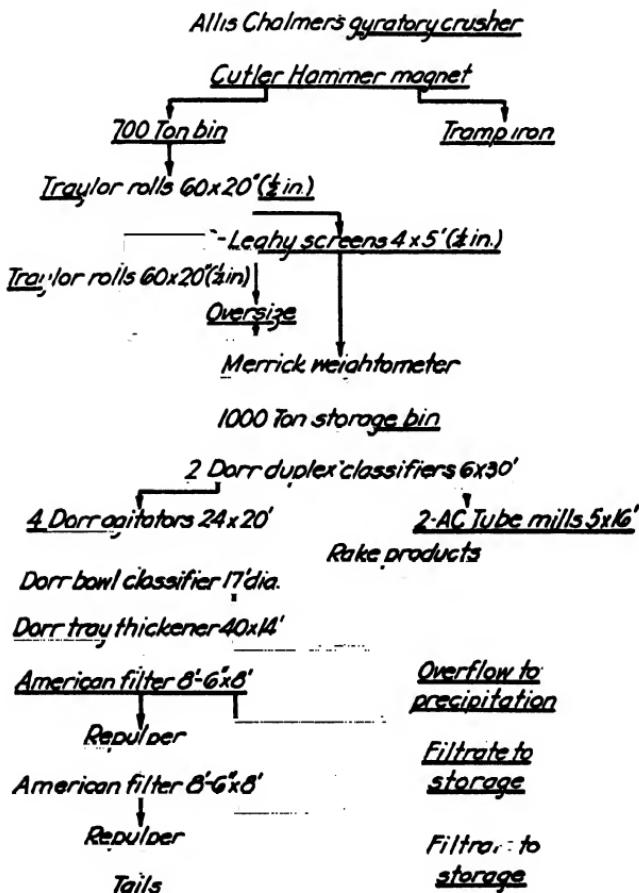


FIG. 99.—Flow-sheet, Coniaurum mill.

\$20.67 per oz. Total recoveries of 95 per cent are obtained with a tailing assaying 23 cts.

Because of the reduced tonnage treated, an excess of crushing capacity permits comparatively fine feed to the grinding mills. This, in turn, permits grinding to 70 per cent minus 200 mesh at

comparatively low grinding costs. Ample storage bins make it possible for the crushers to be operated at one period and the rolls at another, thus conserving power, which is purchased on a peak-load basis.

CRUSHING.—The mine ore is broken in a fine-reduction gyratory crusher, driven by a 100-hp. motor and direct drive, from 10 to $1\frac{1}{2}$ in. The product from this gyratory is conveyed by a 24-in. conveyor and a 20-in. bucket elevator to a 700-ton storage bin, from which the ore is fed to a set of 60- by 20-in. Traylor rolls, running at 1000 ft. per min., peripheral speed. The rolls are driven by two 35-hp. motors with belt drives. The product from the rolls is conveyed by a 24-in. belt conveyor to two Leahy 4- by 5-ft. vibrating screens, using $\frac{1}{4}$ -in. screen cloth, and the oversize is fed to a second set of 60- by 20-in. Traylor rolls running at 1350 ft. per min. The reduction in this step is from $\frac{1}{2}$ to $\frac{1}{4}$ in. The product from these secondary rolls returns to join the feed of the vibrating screen, and a circulating load of approximately 60 tons per hour is maintained in this part of the crushing circuit. The screened product is conveyed by a 24-in. belt conveyor to a 1000-ton ore bin.

In a crushing shift of 11 hr., 470 tons of ore is crushed to minus $\frac{1}{4}$ in. from run-of-mine ore. Wear on manganese roll shells amounts to 0.114 lb. per ton ore crushed, and the cost of coarse crushing, screening, conveying and delivery to the mill ore bins amounts to 24.9 cts. per ton ore milled.

GRINDING.—Grinding is done by a 5- by 16-ft. Allis-Chalmers tube mill, which is in closed circuit with a 6- by 30-ft. Dorr classifier. The classifier overflow at 1 to 1 dilution is pumped by a 4-in. Wilfley sand pump straight to the agitation tanks. Approximately 75 per cent of the total value in the heads is extracted in the grinding circuit.

Screen analysis on final grinding shows 70 per cent minus 200 mesh. Tube-mill ball consumption amounts to 1.7 lb. per ton ore ground. Pocket-type liners are used, and the balls are retained in the pockets, taking some of the wear, which is undetermined. Liner wear is 0.0975 lb. per ton ore ground.

Grinding and classification costs are 17.5 cts. per ton of ore milled.

AGITATION AND THICKENING.—Agitation at 1.25 to 1 dilution for 18 hr. is done in four 24- by 20-ft. Dorr agitators, two unites in

tandem. The agitator discharge goes into a Dorr bowl classifier, which acts as a concentrator as well as a classifier, materially reducing the value of the feed to the thickeners. This concentrated product of the bowl classifier is returned to the tube mill for further grinding, as it contains approximately \$3.75 in gold, whereas the agitator pulp contains only approximately 60 cts. (gold at \$20.67 per oz.), thus showing that there is a concentration of gold in the coarse product from the bowl classifier. The overflow from the bowl classifier is pumped to a 40-ft. Dorr tray thickener. Dissolution of gold during agitation and thickening amounts to 16 per cent of the total in the heads.

FILTRATION.—Two American eight-disc filters, 8 ft. 6 in. in diameter, are used in series with repulping between filtrations. Barren-solution wash is used on the first filter, and water on the second. The final cake from the secondary filter is sent to waste. About 4.0 per cent of the original value in the heads is extracted during filtration. The filter cake from the primary filter is repulped with barren solution and is sent to a 16-ft. diameter by 20-ft. deep Dorr agitator, where an additional 4 hr. agitation contact time is obtained at a pulp dilution of $1\frac{1}{2}$ to 1. The pulp from this agitator is fed to the secondary American filter by means of a 6-in. air lift. The tailing shows an undissolved loss of 20 cts. and soluble loss of 3 cts. per ton. Recoveries in various stages of the milling process are as follows:

Grinding circuit, 75 per cent extraction; agitator circuit, 16; filters, 4; total, 95.

CLARIFICATION.—The overflow from the thickener is pumped to a Hardinge sand filter for clarification. This filter, clarifying 700 tons solution per day, gives a clear solution preparatory to precipitation. Precipitation is done by standard Crowe vacuum system using zinc dust. Reagent consumption, in pounds per ton ore milled, is: Cyanide, KCN, 0.5 lb.; lime, 2.1; zinc dust, 0.0205; lead acetate, 0.0075. In milling 400 tons ore per 24 hr., as in 1933, only one tube mill and two of the four agitators are used.

Dome Mines.—To P. D. P. Hamilton's article, previously cited, we are indebted for the following details of operations of the Dome mill. Since its discovery in 1909, the property has passed through a period when it was considered that the life of the mine was over; has twice survived the calamity of having its

milling plant completely destroyed by fire and is now one of the most important gold producers in Canada.

Metallurgical problems have been very serious at times, but these have all been overcome, with the result that the mill is now



FIG. 100.—Beattie Mines, Ltd., Quebec. Some of the largest and heaviest grinding and classifying units in Canada operate in this flotation and cyanidation mill.

making exceptionally fine metallurgical extractions. Blanket practice has been developed to a high state of efficiency in the present mill. The description of this part of the flow-sheet is given in the chapter "Concentration."

Figure 101 shows the flow-sheet of the crushing plant and mill. The run-of-mine ore is crushed in a Farrell jaw crusher to

4-in. size and is then conveyed to a $5\frac{1}{2}$ -ft. standard Symons cone crusher, which is set for a $\frac{3}{8}$ -in. product. This product is passed over Hum-mer vibrating screens with $\frac{3}{8}$ - by $\frac{5}{8}$ -in. openings. The oversize goes to 18- by 42-in. rolls set at $\frac{1}{4}$ in., and this product

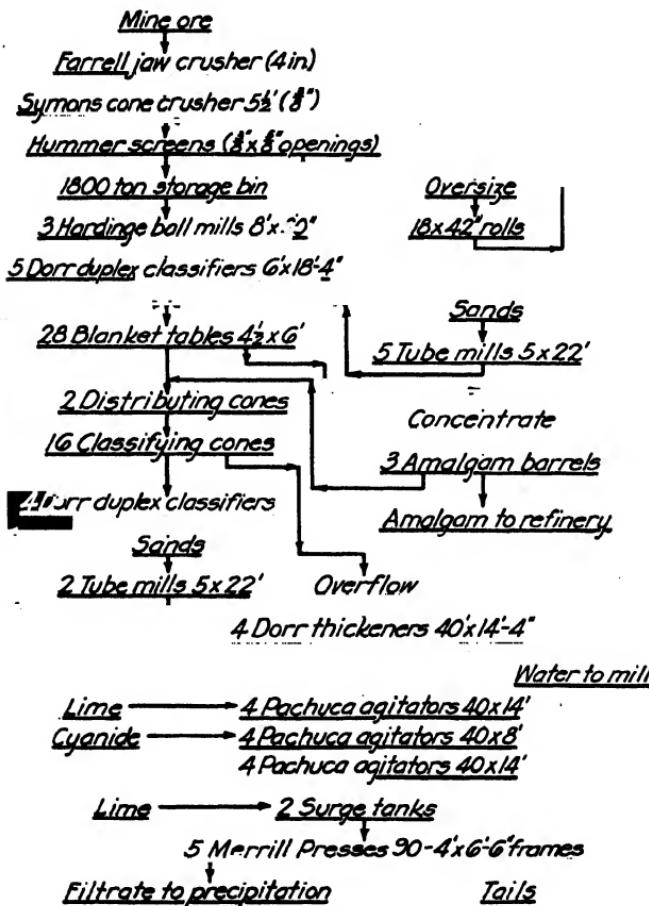


FIG. 101.—Flow-sheet, Dome mill.

returns to the Hum-mer screens. The undersize from these screens is delivered to the mill bin by a belt conveyor. Ore from the mill bin is fed by conveyors to three 8-ft. by 30-in. Hardinge ball mills, using 4-in. balls. The discharge from these goes to five Duplex D 6- by 18-ft. 4-in. Dorr classifiers, which in turn feed five 5- by 22-ft. tube mills which use No. 4 Danish

pebbles. The product from the tube mills and the overflow from the classifiers are pumped to the blanket plant.

The tailings from the blanket plant are pumped to two large cones, which distribute the flow to 16 smaller cone classifiers. The spigot discharge from these cones is pumped to four Duplex C Dorr classifiers. The rake product is fed to two tube mills, 53 in. by 22 ft. inside diameter, which use 1½-in. steel balls. The overflow from the Dorr classifiers and the product from the regrinding tube mills join the flow from the primary grinding units and are pumped to the blanket plant.

The overflow from the 16 cone classifiers flows to four 40-ft. diameter by 14-ft. 4-in. Dorr tray thickeners. The clear-water overflow returns to the general mill circuit, while the underflow is pumped to four 40-ft. diameter by 14-ft. Pachuca agitators, where lime is added, and the pulp is agitated for about 7½ hr. This preliminary agitation increases cyanide extraction and reduces cyanide consumption. The aerated pulp then has cyanide solution added to it and is pumped to another series of Pachuca tanks, where it receives about 11½ hr. agitation in cyanide solution. The pulp then discharges into two storage or surge tanks, which in turn feed by gravity to the five Merrill filters with ninety 4-in. frames each. The unclarified solution from these presses is then pumped through Merrill clarifiers, and the gold is precipitated out by standard Merrill-Crowe precipitation. The pulp, discharged from the Merrill slime press, is thickened in Dorr thickeners and pumped into the tailings-storage dam, the thickener overflow being used for sluicing out the Merrill presses.

In the report to shareholders for the year ended Dec. 31, 1933 the following results of the mill operations for the year are given: heads, 8.0303 dwt. per ton; recovery, 7.8881 dwt. per ton or 98.23 per cent. This is an increase from 95.08 per cent at the beginning of the period.

The consumption of cyanide, NaCN, in pounds per ton milled was 0.4; of lime, 2.6 lb.; zinc dust, 0.044; and lead acetate, 0.03.

Hollinger Mill.—This mill, the largest in the Canadian North, treats about 5000 tons per day. A description of this operation is given by P. D. P. Hamilton in *Trans. 112, A.I.M.E.*, p. 624; and in the Hollinger Edition of *C.M.J.*, September, 1935, the various departments of the mine and mill were very completely

described by different members of the Hollinger staff. The following description and data are taken in part from these publications, but through the courtesy of the management the flow-sheets are corrected as of July, 1936, and data unless otherwise indicated cover 1935.

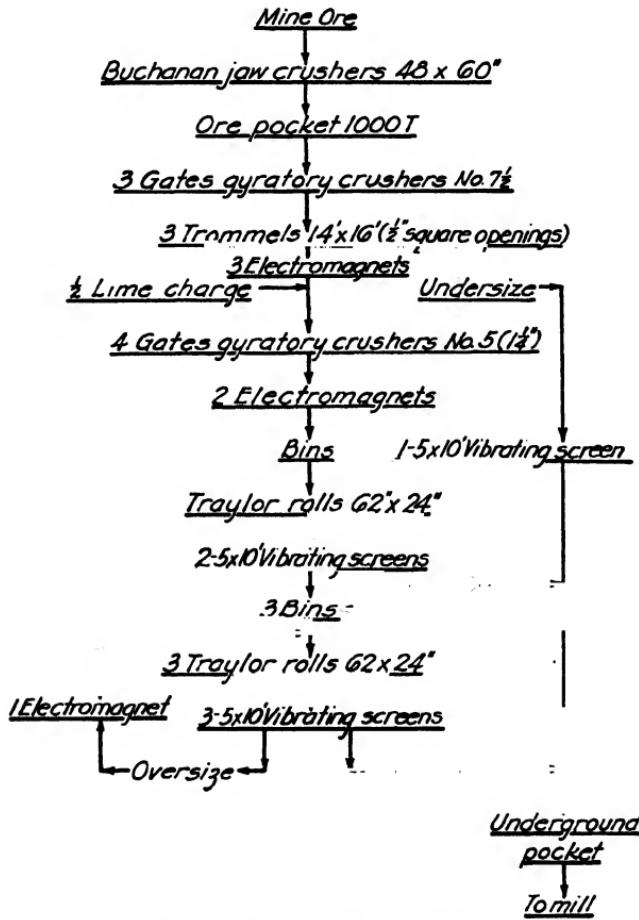


FIG. 102.—Flow-sheet, Hollinger crushing plant.

In the last few years the mill heads have averaged somewhat over \$5 in gold, with gold at \$20.67 per ounce. The ore is described as siliceous schist containing about 4.2 per cent pyrite, of medium hardness and specific gravity 2.8.

In brief, the general plan of treatment after crushing consists of grinding in cyanide solution, table concentration of the sulphides, concentrate regrind with extra concentrate agitation, combined agitation of table tails and concentrate and two-stage filtration.

CRUSHING.—Figure 102 shows the new 1936 flow-sheet of the crushing plant. Primary crushing is carried out underground in a 48- by 60-in. Buchanan jaw crusher set to 6 in., and the crushed ore is hoisted to the surface in skips. The ore is dumped directly from the skips to three No. $7\frac{1}{2}$ Gates gyratory crushers set at $2\frac{1}{2}$ in. The crusher product is sized in three 4-ft. diameter by 16-ft. long trommels with $\frac{1}{2}$ -in. square mesh wire screen. The trommel undersize goes to a vibrating screen, and the oversize drops to four No. 5 Gates gyratory crushers set to crush through $1\frac{1}{4}$ in. The crusher product then goes to one or two 62-by 24-in. Traylor rolls set at $\frac{5}{8}$ in. The product from these rolls goes to one 5- by 10-ft. vibrating screen with $\frac{1}{4}$ - by $\frac{1}{2}$ -in. openings. The screen oversize is given a final crushing in two or three 62- by 24-in. Traylor rolls, set at $3\frac{1}{6}$ in., each of which is in closed circuit with one 5- by 10-ft. vibrating screen with $\frac{1}{4}$ - by $\frac{1}{2}$ -in. openings.

The combined undersize from all the screens is conveyed to an underground storage pocket with a drawoff capacity of 1200 tons and is then fed by feed rollers into weighing hoppers in 7-ton lots. The hoppers are dumped into 7-ton ore cars and transported by cable up a $\frac{1}{4}$ -mile inclined track to the mill bins.

**CRUSHING, SCREENING AND CONVEYING COSTS AT HOLLINGER MILL, 1935
(Cents per Ton)**

Coarse and fine crushing	Cents	Screening and conveying, dry-crushing plant	Cents
Jaw crusher.....	0.68	Screening.....	0.80
No. $7\frac{1}{2}$ gyratory.....	0.81	Conveying.....	0.82
No. 5 gyratory.....	0.95	Dust control.....	0.21
All rolls.....	1.80	Conveying to mill ore bins..	2.28
Total.....	4.24	Total.....	4.11
Total cost of crushing ore and conveying to mill bins.....			8.35 cts.

The above table shows approximate crushing, screening and conveying costs in the early part of 1936 when the finished

mill feed was about 5.0 per cent plus $\frac{1}{4}$ in., 45 per cent minus 8 mesh and 15 per cent minus 200 mesh.

GRINDING.—Figure 103 shows the flow-sheet of the grinding plant and mill. The mill feed is fed from the ore bins to each

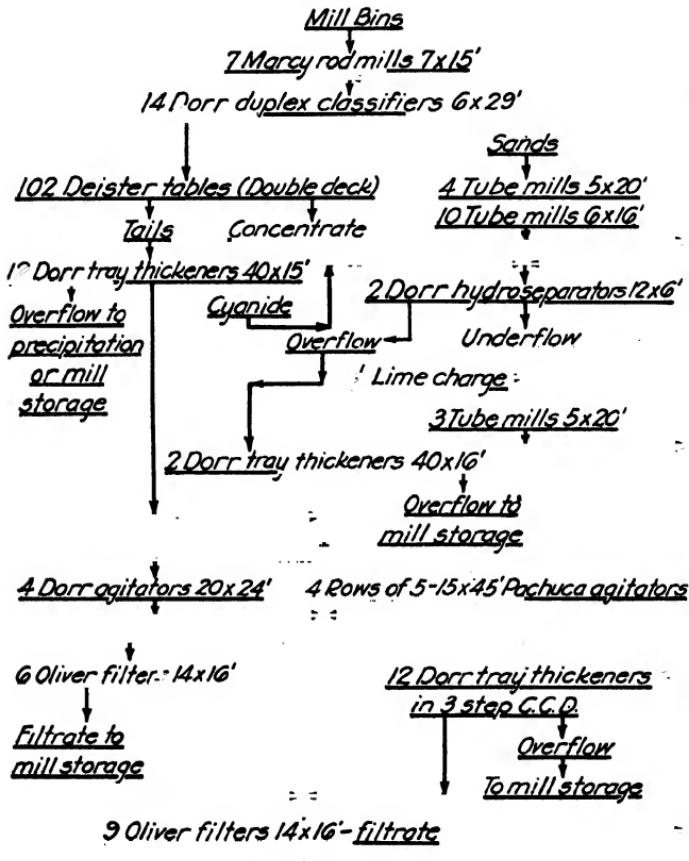


FIG. 103.—Flow-sheet, Hollinger mill.

grinding section by five variable-speed rollers. Primary grinding is done in seven 7- by 15-ft. Marcy rod mills in open circuit. The primary mill discharge goes to fourteen 6- by 29-ft. Dorr duplex classifiers in closed circuit with ten 6- by 16-ft. and four 5- by 20-ft. tube mills. The rod mills have 200 connected horsepower, and the tube mills 125.

Late in 1935 one of the grinding sections was changed to single stage by converting the rod mill to a ball mill with a 12-ft. FX Dorr classifier in closed circuit, and the tube mills were discontinued.

The daily tonnage through this unit at present averages 1275 tons. Experimental work is still in progress in this unit, but results to July, 1936, indicate economies over the older flowsheet, and a cost of under $10^{1\frac{1}{2}}$ cts. per hour is anticipated.

Grinding costs for 1935 follow. During this period mill feed averaged 11.3 per cent plus $\frac{3}{8}$ in. and minus $\frac{1}{2}$ in.

	Per Cent
Rod mill.....	7.29
Tube mills.....	5.98
Classification.....	1.07
Cleaning up, conveying, spillage, etc.....	0.94
 Total.....	 15.28

Extraction in the grinding circuit is about 45 per cent head values when grinding to 2 per cent plus 48 mesh with 65 per cent minus 200 mesh.

Rod-mill steel consumption using 3-in. diameter rods averages about 0.70 per ton milled. French-flint consumption in the tube mills is about 1.80 lb. per ton milled. Liner wear is as follows: rod mill (smooth, manganese steel), 0.165; tube mill (El Oro white iron), 0.086 lb. per ton milled.

CONCENTRATION AND CONCENTRATE TREATMENT.—The overflow from the Dorr classifiers is pumped by 8-in. Morris pumps to 112 double-deck Deister slime-concentrating tables where 70 per cent of the pyrite and 85 per cent of the gold in the table feed is removed. About 800 tons concentrate is produced from 5000 tons feed, giving a ratio of concentration of approximately $6\frac{1}{4}$ to 1. This concentrate is given a further grind to set free gold that is finely disseminated throughout the sulphides. This is done with three 5- by 20-ft. tube mills loaded with $1\frac{3}{8}$ -in. steel balls in closed circuit with two 12- by 6-ft. deep Dorr hydroseparators. The hydroseparator overflows at an over-all fineness of about 98 per cent minus 200 mesh and 90 per cent minus 325.

The concentrates are thickened in two 40- by 16-ft. Dorr tray thickeners, the overflow going to the mill circuit, and the under-

flow, at 60 per cent solids, going to six 20-ft. diameter by 24-ft. Dorr agitators arranged in series of three. Agitation time is about 38 hr.

About one-half the lime used is added at the head of the regrinding tube mills, and all of the cyanide is added ahead of the hydro-separators. As a consequence, the solution in the concentrate section runs higher in cyanide and lime than the main-mill solution. The solution at the head of the concentrate agitators titrates 1.4 lb. NaCN and 1.3 lb. CaO per ton as against 0.75 lb. NaCN and 0.75 lb. CaO per ton for the main-mill solutions.

The agitator discharge goes to the sump taking the primary thickener underflow and is given further treatment along with the thickened table tails.

Steel consumption in the concentrate regrinding tube mills averages about 0.45 lb. per ton ore milled. Representative cost figures for this part of the operation, per ton milled, are: tabling, 2.5 cts., regrinding concentrates, 2.5 cts., pumping and agitation, 0.93 cents.

TABLE-TAILS TREATMENT.—The table tails are thickened in twelve 40- by 15-ft. deep Dorr tray thickeners. Sufficient overflow is sent to precipitation so as to maintain a precipitation ratio of about 0.99 to 1, and the rest joins the mill circulating solution. The underflow at about 54 per cent solids is pumped to four rows of five 15-ft. diameter by 45-ft. Pachuca and two series of two 20-ft. diameter by 24-ft. Dorr agitators. Agitation time is 15 to 18 hr.

The agitator discharge is split two ways, part going to six primary 14- by 16-ft. Oliver filters, and the remainder to three 40-ft. Dorr tray thickeners in C.C.D. The primary-filter discharge is repulped with barren solution, and this combined with the thickener underflow is filtered on nine secondary 14- by 16-ft. Oliver filters.

The filtrate from the primary filters goes to mill storage, while the secondary filtrate is used as wash water in the decantation thickeners. The secondary filters have a water and a barren-solution wash, and the primary filters have only barren solution.

Tailings losses amount to about 20 cts. per ton, of which about 1.7 cts. is solution loss.

Total mill costs in 1935, including crushing, were 64.7 cts. per ton milled. (See chapter on costs for details.)

PRECIPITATION AND REFINING.—Two 36-in. and one 42-in. Merrill filter presses are used for clarification, and precipitation is by the Merrill-Crowe process. Precipitation ratio is 0.99 to 1 which gives pregnant solution assaying about 4.78 per ton. The details of Hollinger refining are covered in the chapter on bullion recovery.

CONSUMPTION OF CHEMICALS, 1935

	Pounds
Lead acetate per ton solution precipitated	0.0085
Zinc dust per ton solution precipitated.....	0.0454
NaCN per ton ore milled.....	0.470
CaO per ton ore milled.....	2.11

McIntyre-Porcupine Mill.—The McIntyre mill in the Porcupine district presents a somewhat different type of flow-sheet from that of Hollinger, in that a finished tailing is made by flotation concentration.

This operation has been very completely described in a paper by J. J. Denny in the November, 1933, issue of *E. and M. J.* and a paper by P. D. P. Hamilton, *Trans. 112, A.I.M.E.*, p. 630. The following is taken in part from these papers and in part from information supplied through the courtesy of the management to bring it up to date as of July, 1936.

CHEMICAL ANALYSES, MCINTYRE-PORCUPINE

Constituent	Mill-feed average for 14 months, per cent	Flotation- tailings average for 6 months, per cent
Iron pyrite.....	8.21	0.10
Silica.....	54.92	58.28
Combined water and carbon dioxide.....	4.32	9.10
Ferric oxide.....	6.28	6.79
Aluminum oxide.....	14.18	15.07
Calcium oxide.....	4.90	5.61
Magnesium oxide.....	4.33	2.82
Sodium and potassium oxide.....	2.85	2.95
Gold, per ton.....	\$7.70	24 cts.
Specific gravity.....	Dry ore 2.84	Dry tailings 2.72

McIntyre ore consists of quartz, porphyry and schistose basalt and dacite with a pyrite content of 3 to 15 per cent. The gold is associated with both the quartz and the sulphides, predominating in the latter. A typical analysis of the ore fed to the mill over a period of 14 months is given in the table on page 313. The mill at present handles 2400 tons per day.

The general scheme of treatment consists of crushing the run-of-mine ore in jaw and cone crushers to 1 in. and reducing the

Mine ore

Taylor jaw crusher 36x48" (underground)

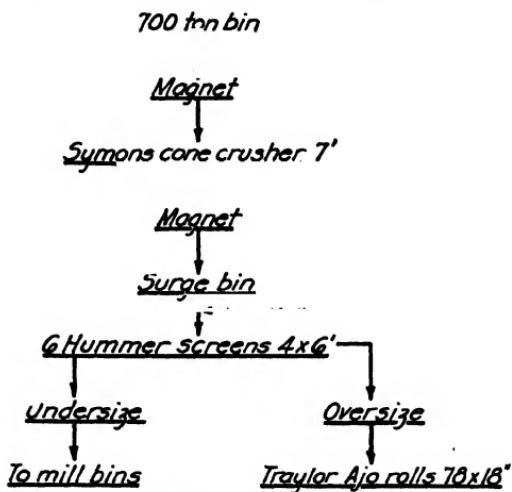


FIG. 104.—Flow-sheet, McIntyre crushing plant.

crusher product to $3\frac{1}{2}$ in. by rolls in closed circuit with vibrating screens. The screen product is ground to 8 per cent plus 65 mesh in tube mills operating in closed circuit with single flotation cells and classifiers. Classifier overflow is floated, with elimination of a finished tailing. Concentrate from the flotation cells is reground in cyanide solution to minus 325 mesh in tube mills, operating in closed circuit with classifiers. Next comes agitation of the pulp and thickening, followed by three stages of filtering and washing, with agitation between stages. Final residue goes to waste. Precipitation of the pregnant solution is done by the Merrill-Crowe system.

CRUSHING.—Figure 104 shows the flow-sheet of the McIntyre-Porcupine crushing plant. Initial breaking is done underground in a 36- by 48-in. Traylor jaw crusher fed by a 64-in. Ross chain feeder. The ore drops into a loading pocket from which it is hoisted to the surface in 6-ton skips which dump into a 750-ton steel storage bin above the cone crusher. Primary crushing is done in a 7-ft. Symons cone crusher set at $7\frac{1}{16}$ -in. which reduces the ore to approximately 1 in. at a rate of 150 tons per hr. Manganese-liner wear is at the rate of 0.020 lb. per ton ore crushed.

The secondary crushing circuit consists of a 190-ton surge bin, six 4- by 6-ft. single-deck Hummer screens and a set of Traylor (Ajo-type) rolls, 78 in. in diameter by 18 in. wide, all in closed circuit; circulating load is 190 per cent. The final screened product or mill feed is all $-3\frac{1}{16}$ in. A complete set of screen analyses of crushing-plant products is given in the chapter on crushing and grinding. Chrome roll shells last 90 days. Dust is kept down in the crushing plant by a Sly collecting system which removes 5 tons dust per day and assays \$2 per ton.

Crushing costs, including labor, employees' insurance, supplies, maintenance, power and general expenses, are given in the accompanying table.

CRUSHING COST, MCINTYRE-PORCUPINE MILL.
(Apr. 1, 1935, to Mar. 31, 1936)

Item	Wear of crusher parts, pounds per ton	Item	Crushing costs, cents per ton	
Jaw plates.....	First set not worn out (3 years)	Symons cone crusher.....	2.37	
Cone crusher.....	0.019	Rolls.....	3.47	
Roll shells.....	0.094	Screens.....	.75	
			Conveyors.....	1.99
			Heating and lighting.....	.36
			Supervision.....	1.43
			Total cost per ton... 10.37	

GRINDING AND CONCENTRATION.—Figure 105 gives the flow-sheet of the grinding and flotation plant. The fine-crusher

product is ground in single stage in five Allis-Chalmers 5-by 16-ft. tube mills. The grinding is to about 8 per cent on 65 mesh. Each mill is operated in closed circuit with a single No. 500 Denver Sub-A flotation cell and a 6- by 30-ft. Dorr duplex classifier. The mill discharges are fitted with 3-mesh screens;

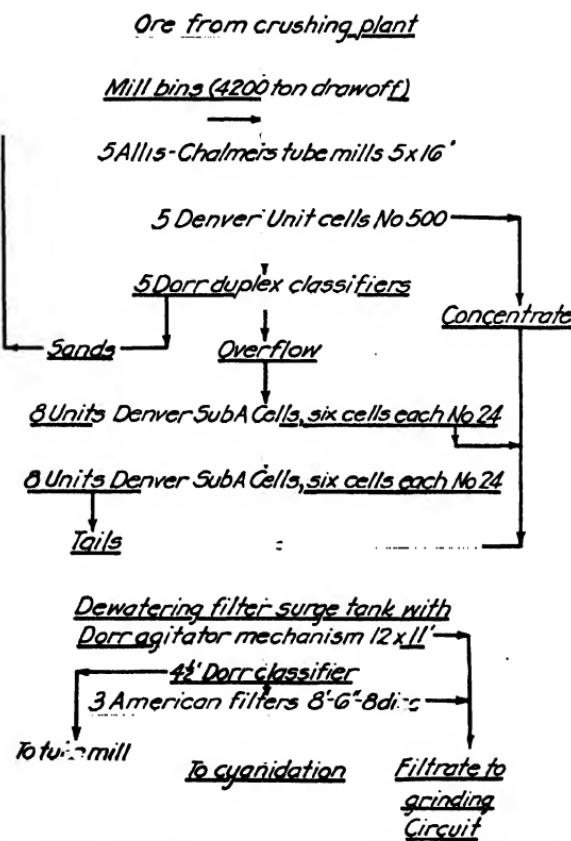


FIG. 105.—Flow-sheet, McIntyre grinding and flotation plant.

the oversize goes directly to the classifier and the undersize to the cells and then to the classifiers.

About 75 per cent of the gold is recovered in the unit cells, 60 per cent floated and 15 per cent removed every 24 hr. from a cone on the bottom of the cell. These unit cells are built with a small hydraulic cone in the bottom to trap gold that is too coarse

to float. This prevents a building up of values in the tube-mill circuit. Hydraulic water added at the bottom dilutes the pulp in the cell and cleans the concentrate therein. Concentrates removed from the cone are added to a concentrate-regrinding tube mill without any detrimental effect.

The main flotation section consists of 48 No. 24 Denver Sub-A primary cells arranged in eight units of six cells each. The primary flotation tailings go to six similar banks of cells. About 98.7 per cent of the pyrite is recovered in the concentrate. The concentration ratio is about 10 to 1 pH in the flotation circuit and varies from 8.9 in the tube-mill discharge to 8.4 in the secondary-cell discharge.

About 50 tons tailings per 24 hr. is pumped to a 15-ft. Wilfley table and serves as a visual guide to the flotation operators. Flotation promoters are added to the tube-mill circuit and frother at both the tube-mill circuit and primary and secondary circuits. Reagent consumption is: collector, 0.12 lb.; frother, 0.08 lb. per ton feed.

Concentrates go to a 12- by 11-ft. filter surge tank equipped with Dorr agitator mechanism where it is thickened to an average dilution of about 1.7 to 1 by twelve 6- by $3\frac{1}{2}$ -ft. clarifier leaves.

To avoid wear by coarse concentrate in the filters, two products were drawn off from the agitator, the coarse from a cone at the bottom and the fine from near the top of the tank. The coarse product now goes to a $4\frac{1}{2}$ -ft. Dorr classifier, making a sand product which is sent direct to the tube mills and an overflow joining the fine product from the agitator and going to three $8\frac{1}{2}$ - by 8-ft. American filters where it is dewatered to 20 per cent moisture and repulped with barren solution.

CYANIDATION.—Figure 106 shows the flow-sheet of the concentrate cyanide plant which has a capacity of 400 tons per day.

Concentrates are reground in a 5- by 16-ft. Allis-Chalmers tube mill in closed circuit with a 6- by 30-ft. Dorr duplex classifier. This mill is rubber lined and loaded with 30,000 lb. $1\frac{1}{4}$ -in. forged-steel balls.

Classifier overflow at 8 to 1 dilution analyzes 97 per cent minus 200-mesh and 88 per cent minus 325 mesh. This overflow combined with overflow from the second regrinding mill is agitated in four 24- by 20-ft. Dorr agitators.

Final agitator discharge goes to a 20-ft. diameter Dorr bowl classifier. This classifier oversize is sent to the second 5- by 16-ft. regrinding mill in closed circuit with a 6- by 30-ft. Dorr duplex classifier.

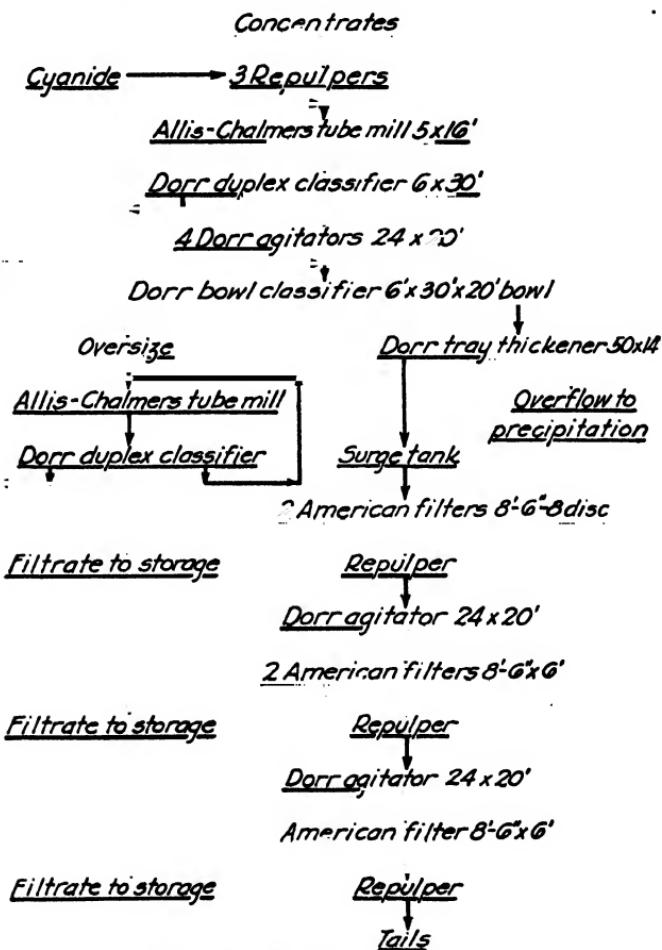


FIG. 106.—Flow-sheet, McIntyre cyanide plant.

The bowl overflow at a dilution of 8 to 1 is sent to a 50- by 14-ft. Dorr tray thickener. This overflow goes to precipitation. The thickened underflow is given three stages of filtration with about 20 hr. of agitation in 24- by 20-ft. Dorr agitators between

the first and second and second and third stages, respectively. Filtration is on five 8.5-ft. diameter, 8-disc American filters.

Water wash on the first filtration step has been found to assist extraction materially. Barren solution is used on the last two stages.

Periods occur when carbonaceous material is present in the ore and is passed to the cyanide circuit. This condition is overcome by diverting the washed residue to two six-cell Denver Sub-A flotation units before discharging it to waste. Approximately 95 per cent of the material is depressed, and a concentrate is recovered that is returned to the cyanide circuit for further dissolution.

About 1950 tons solution per day is clarified and precipitated by the Merrill-Crowe system. Approximately 60 lb. barren solution per ton original mill feed is discarded per day to balance the addition of wash water on the first-stage filters, and this is also sufficient discard to avoid solution fouling. Reagent consumption per ton ore is: cyanide, 0.628 lb.; lime, 1.106; zinc dust, 0.084; lead acetate, 0.012.

Solution strength is maintained at 3.0 lb. cyanide per ton and 0.56 lb. per ton lime. Typical metallurgical results and percentage of recovery are given in the accompanying table.

METALLURGICAL RESULTS AND PERCENTAGES OF RECOVERY

Metallurgical results	Value	Total and stage recovery, per cent
Heads.....	\$7.61	Flotation:
Flotation tails.....	0.253	Unit cells..... 76.00
Cyanide residue.....	0.849	Flotation circuit..... 21.06
Soluble loss.....	0.010	Total..... 97.06
Combined tails.....	0.319	Cyanidation:
		Primary grinding..... 65.00
Gold at \$20.67 per oz.		Agitation and regrind..... 31.50
		Filters, etc. 2.20
		Total..... 98.70
		Total over-all recovery..... 95.80

The foregoing data are representative of results during the first half of 1933. Since that time with the increase in the price of gold and increased tonnage, heads assay in ounces of gold has decreased, and the tailing decreased proportionately. In addi-

tion, cyanide residues have been reduced approximately 20 cts. per ton owing to a water wash on the first filters.

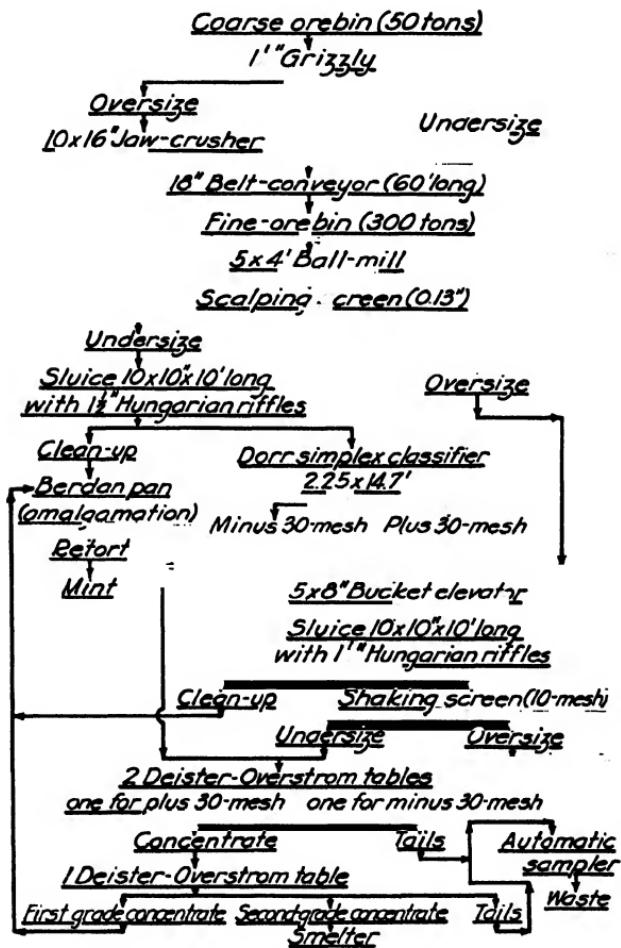


Fig. 107.—Flow-sheet of Original Sixteen-to-one mill, California.

ORIGINAL SIXTEEN-TO-ONE MINE

At Allegheny, Sierra County, Calif., this company mines quartz containing free gold and gold in pyrite and arsenopyrite. The ore carries a little galena and sphalerite. Rich ore is sorted out underground and treated separately by crushing and grinding

and amalgamating in Berdan pans. This rich ore assays 50 oz. and more. The tailings are shipped to a smelter. The reject from sorting constitutes the mill supply and assays 0.5 oz. gold per ton. It is treated at the rate of 100 tons per day, as shown in the flow-sheet in Fig. 107. The first-grade concentrates are amalgamated prior to shipment, and the second grade are shipped direct at the rate of 6 tons per month. Tailings average 0.015 oz.

PIONEER MILL

This mill in the Lillooet district of British Columbia treats 400 tons per day in an all-cyaniding plant.

Much of the material here given is from an article by Russell Spry and Paul Schutz in *Trans. 112, A.I.M.E.*

The mill feed consists of quartz carrying small amounts of various sulphides including pyrite, arsenopyrite, pyrrhotite, chalcocite, etc. Country rock is quartz diorite altered to sericite. This sericite slimes readily and is unwelcome in the mill because of its poor settling characteristics. There are two main sections to the mill, one built later than the other.

Crushing.—Ore is sledged underground to pass 5-in. grizzlies. From the receiving bin it is conveyed to a 4- by 5-ft. Hum-mer screen having $\frac{7}{8}$ - by 5-in. openings. The oversize is crushed in a 3-ft. Traylor bell-head crusher, Number 37, set for $\frac{1}{2}$ -in. product, and, combined with the undersize, is conveyed to the mill over a 300-ft. aerial tram. Mill feed runs about 6.9 per cent plus $\frac{7}{8}$ in. and 48.8 per cent plus $\frac{1}{4}$ in.

Grinding.—Figure 108 shows the flow-sheet of the mill. Grinding is done in solution and in two stages. Primary grinding in the first unit is done in a 5- by 4-ft. Allis-Chalmers ball mill in closed circuit with a 4-ft. 6-in. by 21-ft. 8-in. Dorr duplex classifier; in the second unit a No. 64 $\frac{1}{2}$ Marcy ball mill is used in closed circuit with 8-ft. by 23-ft. 4-in. Dorr duplex classifier.

Secondary grinding in the first unit is done with a 5- by 16-ft. Allis-Chalmers tube mill in closed circuit with a 6-ft. wide by 10-ft. diameter Dorr bowl classifier. The primary mills grind to a few per cent on 10 mesh, and the final grind is from 2 to 5 per cent on 100 mesh and 65 to 75 per cent minus 200 mesh.

Total ball consumption is 3.71 lb. per ton, 1.48 lb. in the primary and 2.23 lb. in the secondary. Liner consumption is 0.56 and 0.39 lb. per ton, respectively.

Agitation and Thickening.—The bowl-classifier overflow first goes to thickeners where the pulp is thickened before agitation, and the decanted solution is sent to precipitation.

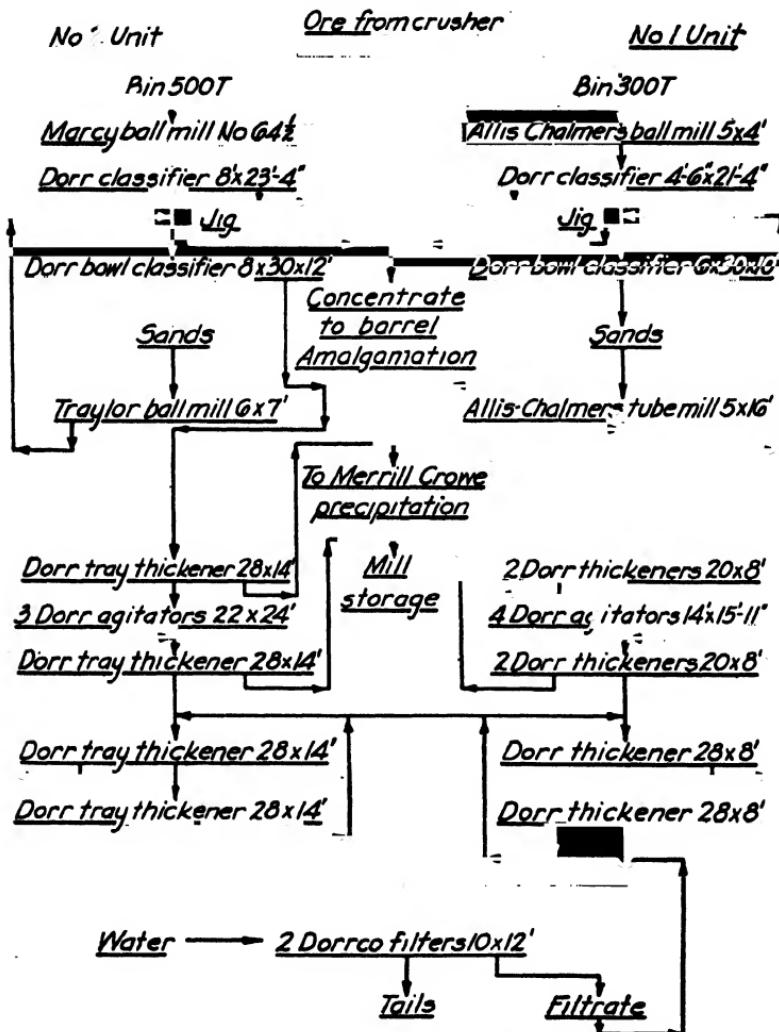


FIG. 108.—Flow-sheet of Pioneer mill.

The primary thickeners are followed by Dorr agitators which, on a basis of 400 tons per day of feed at 50 to 52 per cent solids, provide a 44-hr. treatment period in the series.

Agitation is followed by three stages of C.C.D. in various size Dorr thickeners. Because of the need of greater elevation between the balanced-type thickeners in the second unit, Inspiration-type Dorreco diaphragm pumps are used. In this type of pump the usual solid eccentric rod is replaced by a 4-in. pipe, the pipe acting as an eccentric rod and at the same time as a discharge tube through which the pulp is lifted to a height of $5\frac{1}{2}$ ft. above the pump bowl.

Filtration.—The underflow from both units goes to a steady-head box and thence to two Dorreco filters 10 by 12 ft. The amount of wash water used in the sprays is recorded by meters. Filtrate is usually pumped to the second last thickeners in the washing series.

The following table shows the dissolution at various stages in the mill based on averages for the first 5 months of 1934. Solution loss in the tailings for the same period averaged 0.0023 oz. gold per ton. Dissolution on the filters for this period was 0.001 oz. gold per ton but for the 6-months period preceding was 0.004 oz. gold per ton. Present dissolved loss in the tailing has now reached the low figure of 0.001 oz. gold per ton solids.

DISSOLUTION OF GOLD

Location of sample	Assay (solids), ounces gold per ton	Gold dissolved, per cent	
		Indi- vidual	Cumulative
Mill feed.....	0.7133		
Agitator feed.....	0.2405	66.28	66.28 prior to agitators
Agitator discharge.....	0.0324	29.31	95.59 in agitators
No. 2-thickener underflow.....	0.0227	1.23	96.82 in No. 2 thickener
No. 3-thickener underflow.....	0.0207	0.27	97.09 in No. 3 thickener
No. 4-thickener underflow.....	0.0199	0.12	97.21 in No. 4 thickener
Filter cake to waste.....	0.0189	0.14	97.35 in filter.....

Clarification and Precipitation.—A precipitation ratio of 3.25 to 1 is maintained. Clarification is in a 42- by 42-in. Merrill press, after which the solution goes to a 20- by 8-ft. storage tank. According to Spry and Schutz there is a definite advantage in keeping the pregnant solution above 60°F. during precipitation.

This is done by passing an 8-in. header from the mine compressor through the storage tank. Precipitation is by the Merrill-Crowe process, a Merrill-Crowe, low-level vacuum leaf filter being used. The latter consists of twenty-four 4- by 6-ft. leaves in a 10½- by 12-ft. tank in which an impeller provides agitation. One of these filters is used as a spare to avoid delay during cleanup.

Precipitation and Refining.—Precipitate washed from the precipitation filters is pumped to a Merrill press in the refinery and dried to 8 per cent moisture by blowing the press with low-pressure air.

Smelting and refining are done in a 3- by 4-ft. (inside dimension) Monarch simplex furnace, and bullion is melted in a Monarch tilting crucible furnace. "Carbofrax" lining in the Monarch furnace lasts 110 to 130 hr.

Solution Strengths and Reagent Consumption.—Cyanide is added at the grinding mills and in the first two agitators of each unit. Quicklime is added as the ore is dumped from the skips, and milk of lime is fed from drip boxes to the first secondary thickeners.

Pregnant solution is kept at 1.4 lb. NaCN, and grinding solution about 1.3 lb. NaCN per ton. Cyanide consumption for the first 5 months of 1934 averaged 0.87 lb. per ton ore treated. This figure has been lowered considerably since, present consumption of cyanide (1936) being 0.5 lb. NaCN per ton ore. Pregnant solution is kept at 2.36 lb. CaO, and grinding solution 2.1 lb. CaO per ton. Lime consumption is 4.23 lb. per ton ore. High lime is carried to aid settling in the thickeners.

NOTE.—In early 1936 total recovery had reached 97 per cent and milling costs, including refining, were less than \$1 per ton.

PREMIER TREATMENT

The Premier mine, near Portland Canal, British Columbia, is a good example of dressing a gold-silver ore for eventual treatment of the concentrate. The silver-gold ratio is now about 18 to 1. During 15 years the average was 13 oz. silver and 10 dwt. gold per ton. For several years concentration on Wilfley tables and by flotation, followed by cyanidation, gave an extraction of 87 per cent of the silver and 97 per cent of the gold. In 1926, cyanidation gave way to tabling and flotation and shipment of

the concentrates; and in 1927, all-flotation was adopted. According to W. J. Asselstine and D. L. Coulter in *Bul. C.I.M. and M.*, February, 1933, recoveries were 86.3 per cent of the silver and 93.3 per cent of the gold in 1931; and the total cost of milling, \$1.067 per ton treated. The ore is siliceous and carries, besides the precious metals, some iron pyrite, lead and zinc.

Mill operation and laboratory tests showed that 50 to 60 per cent of the mineral content could be recovered when grinding to 50 per cent through 200 mesh and that the remainder could be taken off as a middling product to be returned to the circuit for further classifying, grinding and floating. By adopting this method, the fine grinding was decreased from 81 to 70 per cent through a 200 mesh without affecting the recoveries or the grade of concentrate. The Premier cell, operating with a cone classifier, was an added development of this plan. It is of the cascade type and was devised primarily for use between the primary grinding and regrinding circuits, also between the regrinding circuits and their respective cone classifiers. These cells remove a coarse concentrate as soon as grinding has reached the stage at which flotation is possible. The capacity of the filters was nearly doubled by this arrangement.

Grinding is done by Marcy and Hardinge ball mills in closed circuit with Dorr classifiers and by Allis-Chalmers and Colorado Iron Works tube mills in circuit with the flotation cells. Another Hardinge mill in circuit with a Dorr classifier does further grinding. The pulp then goes to Minerals Separation and K. and K. flotation machines. Dorreco and Wilfley pumps circulate the pulp. The concentrate is thickened in a Dorr machine, dried in an Oliver filter and shipped to a smelter.

The reagents used amount to 0.03 lb. sodium carbonate, 0.01 lb. sodium cyanide and 0.27 lb. Aerofloat per ton ore milled. The first two are added at the secondary crusher, and the Aerofloat at the fine-grinding mills. The plant handles nearly 470 tons per day.

SPRING HILL TREATMENT

Near Helena, Mont., the Montana Mines Corporation treats 150 tons ore daily, according to L. A. Grant in *I.C. 6411, U. S. B. of M.*, 1931. Most of the gold is associated with pyrite, arsenopyrite and bismuth, but 10 to 15 per cent is free and very fine.

Barren pyrrhotite is the most prominent sulphide as to volume. Cyanidation was not a success.

The crushing plant consists of a 24- by 36-in. Traylor jaw crusher, set at 5 in., a conveyor belt with suspended magnet, a bar grizzly with 1- to 1½-in. openings, a 14- by 18-in. jaw crusher set at 1½ in., a conveyor belt, a 4- by 5-ft. Leahy vibrating screen with $\frac{9}{16}$ - by 1-in. openings, a Symons cone crusher set at $\frac{1}{2}$ in. and a belt conveyor delivering to the mill bin.

Two 7-ft. by 36-in. Hardinge ball mills, with a load of 5 parts circulating and 1 part new feed, in closed circuit with Dorr classifiers, grind the feed so that 59 per cent passes 200 mesh. Flotation reagents amounting to 0.075 lb. pine oil, 0.109 lb. xanthate, 0.103 lb. Aerofloat, and 0.088 lb. copper sulphate are added to the mills.

The classifier overflow is diluted to 19 per cent solids and is fed to 10-cell 24-in. and 12-cell 18-in. Minerals Separation flotation machines. The ratio of concentration is 12 to 1. Tailings are thickened, and the water reclaimed; the concentrates are dried on a two-disc American filter. They assay $3\frac{1}{4}$ oz. gold per ton.

Gold recovery averages 82 per cent at a cost of \$1 per ton. Electric power cost 0.8 ct. per kw.-hr.

UNITED EASTERN MILL

The United Eastern mill is in the Oatman Gold Roads district of Mojave County, Arizona, and was built about twenty years ago. It is included in this volume not as an example of modern practice but rather as exemplary of one of the transition stages in cyanide mill evolution. Capacity was initially 200 tons per day but was later increased to 300 tons.

The following description of this mill and accompanying flow-sheet (Fig. 109) are from two very complete papers on the construction and operation of this property, by Otto Wartenweiler in *Trans. 59, A.I.M.E.*, 1918, and by Wheeler O. North in *Trans. 63, A.I.M.E.*, 1920.

The ore consists of a mixture of calcite and quartz in about equal proportions, with some decomposed andesite. Average value is calculated to be \$20 or more gold per ton with no silver present.

A No. 6 Telsmith gyratory crusher reduces run-of-mine ore to 3 in. Product passes over a Merrick weightometer on its way to a 24- by 24-ft. circular wood bin ahead of the rotary mills.

Primary grinding is done in a No. 64½ Marcy mill, loaded with 9000 lb. of 5-in. balls, in closed circuit with a 3-ft. wide Simplex Dorr classifier, overflowing at 20 mesh.

Primary classifier overflow is sent to the secondary grinding circuit, consisting of two 5- by 6-ft. Allis-Chalmers ball mills, loaded with 8000 lb. of 2-in. chrome-steel pebs, in closed circuit with two 4-ft. wide Dorr duplex classifiers, overflowing at 82 per cent minus 200 mesh. When capacity was increased, an

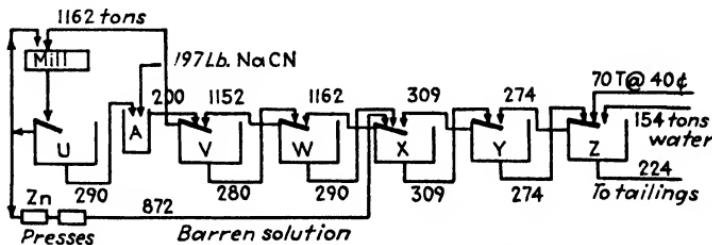


FIG. 109.—Counter-current decantation flow-sheet, United Eastern.

8-ft. Callow cone was placed after each classifier with spigot product returned to the classifier feed box by Frenier pump.

Cone overflow at 4.5 to 1 dilution flows by gravity to a 40- by 12-ft. Dorr thickener. Thickener overflow goes to precipitation while underflow, at a specific gravity of 1.40, is pumped to a series of seven 14- by 24-ft. Dorr agitators. The agitation period is 62 hr. after which dissolving of values is almost negligible.

The product from the last agitator flows through a straight c.e.d. plant, consisting of five 40- by 12-ft. Dorr thickeners and one 30- by 8-ft. Dorr thickener. Underflow is transferred from thickener to thickener by Dorco diaphragm pumps against a counter-flow of 3.76 tons barren solution per ton ore, introduced in thickener 4, and 0.82 ton fresh wash water per ton ore, introduced in thickener 6. At thickener 6, the flow is split, and about one-third is sent to thickener 7, 30 by 8 ft.

The underflows of thickeners 6 and 7 are combined and flow together to a tailings pond at a dilution of 0.82 ton solution per

ton tailings. The overflow from thickener 2, directly after the series of agitators, is returned to the grinding plant as mill solution.

Each day 263 tons ore is crushed in cyanide solution; \$20.66 gold is dissolved as follows:

\$10.00	End of thickener 1
10.30	In agitators
0.20	In thickener 2
0.07	In thickener 3
0.04	In thickener 4
0.02	In thickener 5
0.03	In thickener 6
<hr/>	
\$20.66	

Underflow moisture contents average 51.3 per cent. From thickener 1, 872 tons of pregnant solution are precipitated to

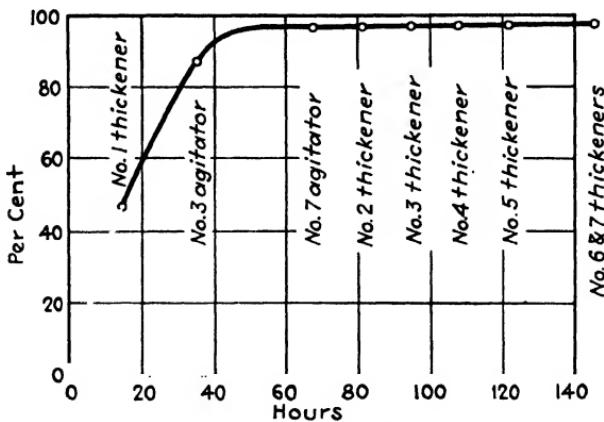


FIG. 110.—Curve of dissolution, 280 tons basis, United Eastern.

\$0.025. Agitation is at 1 part ore to $1\frac{1}{2}$ parts solution. Solution strength is 2.6 lb. KCN per ton.

The theoretical dissolved loss per ton of ore is \$0.1704, the actual \$0.215 and the net \$0.108. The reason for this difference between actual and net dissolved losses was brought out in the discussion of this paper and shown to be due to the fact that 70 tons water a day, impounded at the tailings dam and averaging

\$0.40 gold per ton, is returned to the mill and introduced into thickener 6 as wash. The initial value of the tailings water is appreciably enhanced owing to the additional extraction and evaporation taking place in the tailings pond.

The net loss of \$0.108 is somewhat higher than would normally be expected in such a plant even with such high head values. This is primarily due to crowding capacity from the designed figure of 200 to 263 tons per day with a consequent requirement of raising thickener-discharge moistures from 45 to an average of 51.3 per cent thereby reducing the decantation efficiency.

Precipitation and Refining.—Pregnant solution from thickener 1 is pumped through a Merrill central-sluicing clarifying filter, delivering to a Crowe deaeration vacuum system. Zinc dust is added as the solution leaves the Crowe system on its way to the precipitation presses. Precipitation is 3.3 tons solution per ton ore.

Wet precipitate is fluxed with 11 per cent borax glass, 11½ per cent sodium bicarbonate, 6 per cent manganese dioxide, 3.3 per cent ground bottle glass and at least 10 per cent old slag shells from former melts, all these percentages being on the basis of a dry precipitate. Melting takes place in an oil-fired No. 15 Case tilting furnace, equipped with a long-lipped graphite pot. Each charge yields a 600- to 700-oz. button and some 40 to 50 lb. slag.

Crude, dry precipitate averages 1440 lb. avoirdupois per month, equivalent to 1024 lb. bullion or 8538 troy oz. gold and 4497 oz. silver. Bullion buttons are remelted and cast into 150-lb. bars, having an approximate fineness of 560 gold and 301 silver.

Sodium cyanide consumption averages 0.735 lb. per ton ore, lime consumption 4.55 lb. per ton and zinc dust 0.464 lb. per ton. All cyanide is added to agitator 1 where the cyanide strength is held at 2.7 lb. KCN per ton ore. Lime is added to the primary mill feed.

In 1917, the treatment of 84,548 tons ore, containing 89,163.078 oz. gold and 45,607.16 oz. silver, showed a recovery of 96.36 per cent. In 1918 the corresponding figures were 92,339 tons ore, containing 100,903.079 oz. gold and 54,137.84 oz. silver, and a recovery of 96.95 per cent.

Cost data follow, averaged for 1917 and 1918 operations.

Coarse crushing per ton.....	\$0.0709
Primary grinding per ton.....	0.3171
Secondary grinding per ton.....	0.4985
Cyaniding (agitation and decantation).....	0.6242
Precipitation.....	0.1342
Clarification.....	0.0382
Refining.....	0.1024
Sampling.....	0.0040
Tailings disposal.....	0.0257
Solution heating.....	0.0304
General expense.....	0.1767
Lighting.....	0.0082
Assaying.....	0.0336
Water.....	0.1050
Total.....	\$2.1691

A breakdown of each of the preceding expense items will be found in *Trans. 63, A.I.M.E., 1920, 548-570 inc.*, segregating operating labor, repair labor, supplies, power and miscellaneous.

VALUES OF SOLUTIONS IN VARIOUS THICKENERS, UNITED EASTERN

Thickener	Solution values		Cyanide strength (KCN), pounds per ton	
	Theoretical	Actual	Theoretical	Actual
U	\$6.24	\$6.22	1.90	1.90
V	3.97	5.85	1.9819	2.00
W	1.04	1.78	1.7881	1.80
X	0.29	0.55	1.7407	1.70
Y	0.27	0.40	1.244	1.20
Z	0.20	0.26	0.685	0.80

WITWATERSRAND

Ore mined in the vicinity of Johannesburg, Transvaal, is a conglomerate composed of various sizes of pebbles, principally of barren white quartz, which constitute 70 per cent of the whole. The pebbles are within a matrix of cementing material consisting of fine-grained quartz, sericite, chlorite and chloritoid. The matrix carries the gold, a little silver and the mineral sulphides. Argillaceous components of the mill pulp may be as high as 32 per cent. Of the sulphides, pyrite and pyrrhotite

predominate. Some osmiridium is also present—1 oz. in 3000 to 10,000 tons ore. Most of the oxidized ore has been extracted, so milling is confined mainly to the sulphide ore. This shows by analysis 86.75 per cent silica, 2.75 per cent pyrite, 2.65 per cent ferric oxide, 6.91 per cent alumina and 0.70 per cent magnesia. The specific gravity is 2.7. The gold content ranges from $3\frac{1}{2}$ to $8\frac{1}{2}$ dwt. per ton, and the gold-to-silver ratio 9 to 1. The ore is hard and brittle. Ore ground to 92 per cent through 200 mesh revealed 47 per cent of the gold to be free, 9 per cent encased in gangue and 44 per cent associated with the sulphides. A large portion of the gold in its natural condition is so fine that it is rapidly dissolved in cyanide solution. Fine grinding is necessary.

As showing the development of metallurgical treatment on the Rand the general flow-sheets as given by T. K. Prentice in *Bul. I.M. and M.*, April, 1935, are reproduced in Figs. 111, 112 and 113.

In a general way, according to F. Wartenweiler in *Trans.* 112, *A.I.M.E.*, 1934, the ore of the central and near eastern Rand is amenable to simple treatment; at the extreme west, pyrite and pyrrhotite have been more prominent, and in one "reef" the gold is closely encased in the siliceous matrix. In the far eastern Rand, the gold is associated to a greater extent with mineral sulphides, and finer grinding is required to bring about contact with the cyanide solution. These differences account for the varied treatment given: (1) crushing, stamping, tube milling and separate treatment of sand and slime; and (2) crushing, tube milling and only slime treated. Excerpts from the paper mentioned will be found in this book; a brief general description of the first treatment mentioned follows:

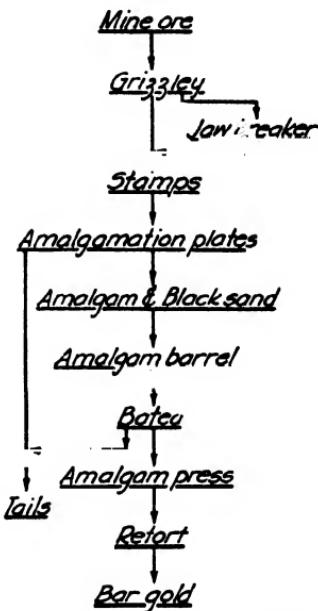


FIG. 111.—Typical flow-sheet, Witwatersrand metallurgical practice, 1886-1890.

Stamp Milling, Sand and Slime Treatment. Crushing.—An average of 15 per cent of the mine rock is rejected by sorters; and of the final crusher feed to the stamps, 36 to 50 per cent is coarser than 1 in., 20 to 24 per cent on $\frac{3}{4}$ in., 11 to 16 per cent on $\frac{1}{2}$ in. and 11 to 20 per cent through $\frac{1}{4}$ in. The stamps crush

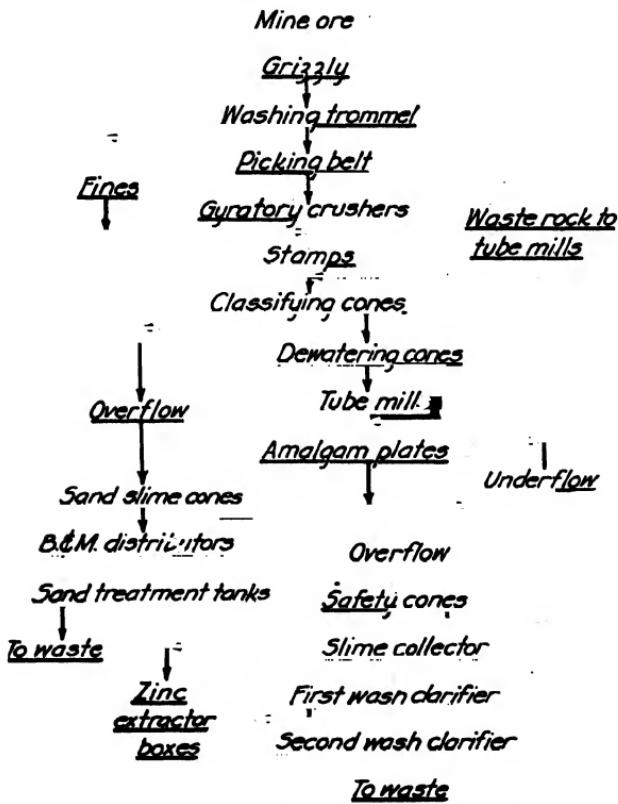


FIG. 112.—Typical flow-sheet, Witwatersrand metallurgical practice, 1910.

12 to 23 tons per day each through coarse screens ranging from 6 mesh per linear in. to 1-in. mesh.

Grinding.—Pulp from the 5603 stamps at work is ground in 416 tube mills which use quartzite as pebbles, in some plants with steel balls. In addition to the ordinary cone, a variety of mechanical classifiers is in circuit with the tubes—the Dorr rake, MacLean spiral and Job cone with the primary mills; and the Dorr bowl machine for secondary and tertiary mills.

Concentrating.—Several methods are employed to handle the sulphide portion of the ore. The concentrating action of classifiers in the tube-mill circuit permits preferential grinding of the sand and pyrite. Corduroy tables or Johnson concentrators

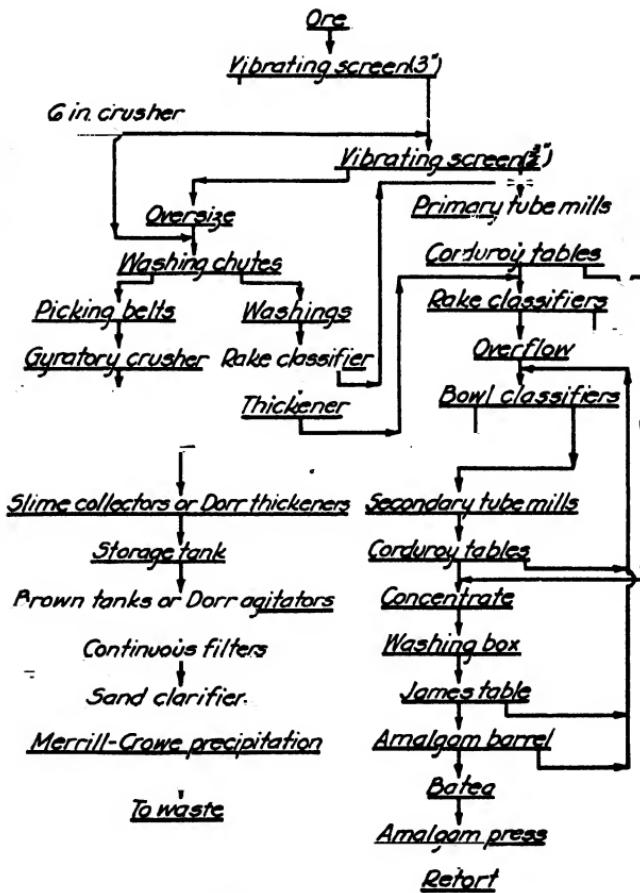


FIG. 113.—Typical flow-sheet, Witwatersrand metallurgical practice, 1934.

are used to save coarse gold and pyrite. The former recovers 25 to 60 per cent of the gold, a concentrate which, after dressing, is 0.05 per cent of the ore tonnage. The Johnson machine, which is a rotating drum, recovers 44 per cent of the pyrite and 55 per cent of the gold in a concentrate weighing 8 per cent of

its feed. After its free gold has been taken out by blanketing, the concentrate is ground and then combined with the cyanide pulp. Another concentrator is the Smith and Maxwell jig, one for each tube mill. The concentrates from this jig are dressed to 2 per cent of the ore milled, ground cyanided and the residue sold to acid works.

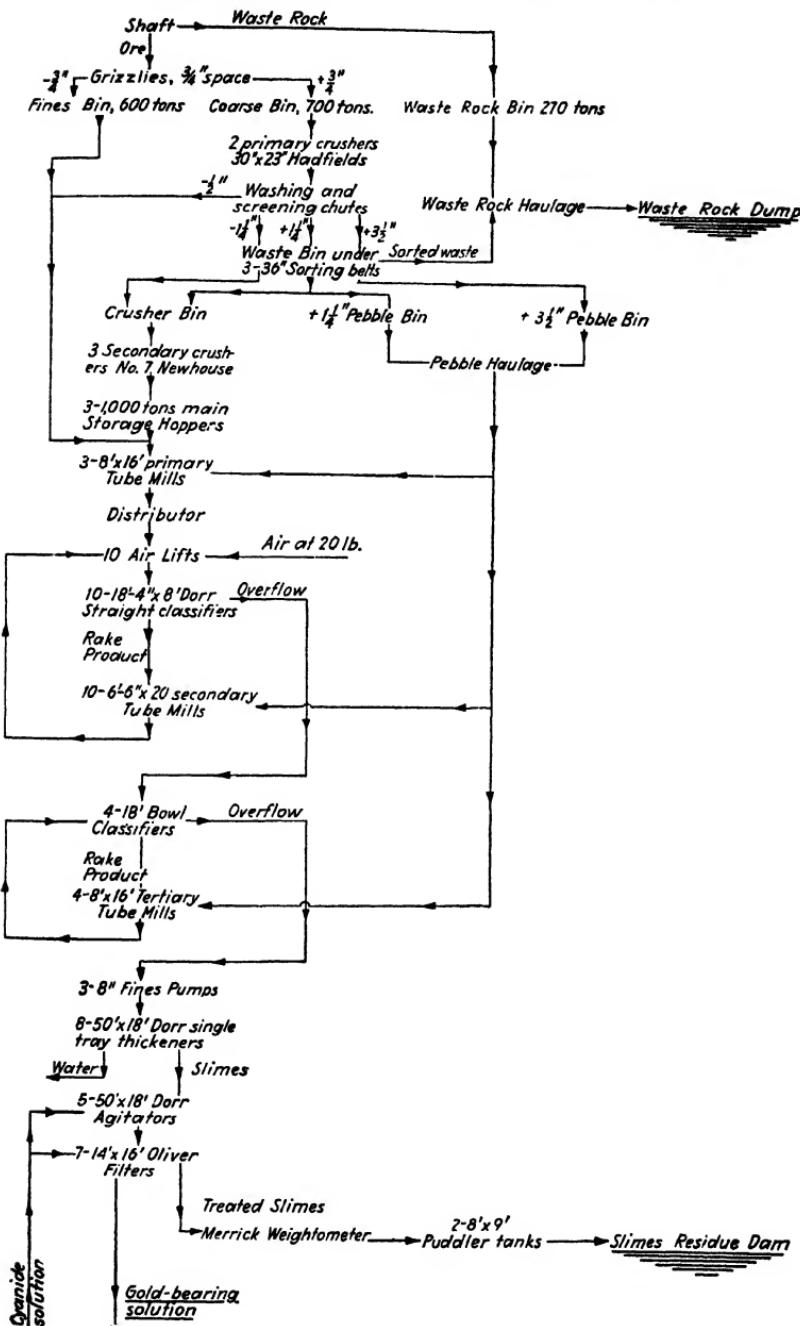
Sand Treatment.—As much as 60 per cent Rand ore was leached, but this percentage is lessening, and in some plants the proportion is as low as 10 per cent. The trend is toward all-slime treatment. All plants, except seven which are all-sliming, treat the sand separately by standard leaching methods.

Sieve tests of sand show the following range of sizes: 26 to 61 per cent through 48 and on 100 mesh, 29 to 56 per cent through 100 and on 200 mesh. In these examples the sand constituted 26 to 68 per cent of the tonnage. The sand is leached 3 to 7 days with cyanide solution, the strongest being 0.06 per cent KCN, with an alkalinity of 0.005 per cent CaO. Water washes are seldom practicable. Aeration of the dissolving solution, as described elsewhere, is receiving attention at most plants. At one works, a small volume of air under pressure is introduced during the first saturation periods, and drainage is assisted by suction during the last two or three periods. This more thoroughly oxidizes the ferrous compounds, lessens the cyanide consumption and shortens the treatment.

Slime Treatment.—Agitation of slime is effected in Brown (Pachuca) tanks or in Dorr machines, in charges or in series, for 6 to 30 hr., with solutions of 0.01 to 0.025 per cent KCN. Butters leaf or Oliver drum filters are used for washing and dewatering. Some decantation plants are in operation.

Precipitation.—Gold is precipitated on both zinc dust and zinc shavings, all of the newer and some of the older plants using the former. The respective consumption of zinc is 0.05 and 0.14 lb. per ton ore milled.

Cost.—The total cost of ore dressing and ore treatment averages 32 pence or 65 cts. per ton ore milled to extract an average of 96 per cent of the gold. At the Central Mining-Rand mines group, in 1934, the cost of crushing, tube-milling and all-slime treatment was 34.2 pence (68.4 cts.) per ton milled, compared with 33.8 pence (67.8 cts.) for crushing, stamping, tube-milling and sand and slime treatment.



East Geduld.—Through E. P. Cowles, general manager of East Geduld mines, Springs, Transvaal, H. A. White, reduction manager, supplied the following information.

Of 3200 tons of "blanket" mined in 24 hr., 400 tons is rejected as waste. Washings from the sorting operation are sent to the tube mills. The ore averages 7 dwt. gold and 0.7 dwt. silver per ton, also 3 per cent pyrite.

Coarse crushing is done by a 24- by 36-in. Hadfield jaw crusher set at 5 in. and fed by a Ross chain feeder, followed by a 7-in. Newhouse crusher set at $\frac{3}{4}$ in. No screens are used. Of the final product, 75 per cent passes $\frac{3}{4}$ in.

Primary grinding is done in three 8- by 16-ft. tube mills in open circuit; secondary grinding in ten $6\frac{1}{2}$ - by 20-ft. tube mills in closed circuit with 8- by 16-ft. Dorr classifiers set at 3 in. to the foot and working at 24 strokes per min.; and tertiary grinding in four 8- by 16-ft. tube mills in closed circuit with 20-ft. Dorr bowl classifiers. Mine quartzite serves as grinding pebbles. The final overflow is 90 per cent through 200 mesh. All grinding is in water.

The slime is thickened in eight Dorr two-compartment tray thickeners, 50 by 18 ft. Dorco quadruplex pumps draw off the slime with 45 per cent moisture and send it to five Dorr agitators of the same size as the thickeners. The agitators consume 4 hp. each and use air at 20 lb. pressure. The strength of the solution is 0.5 lb. KCN per ton solution. The consumption of cyanide is 0.35 lb. and of lime 2 lb. per ton ore. The contact period in the agitators is 26 hr. Filtration is done on seven 14- by 16-ft. Oliver machines, the cake of which contains 26 per cent moisture. The loss of dissolved gold in the final residue is 0.01 dwt. Lead acetate is added to the agitators and to the solution before it is clarified, the total being 0.05 lb. per ton.

Precipitation of the gold and silver from 4480 tons solution per day is by the Merrill-Crowe process, using 0.06 lb. zinc per ton ore treated. Solutions are deaerated before precipitation and aerated before use in the plant. The loss of undissolved gold is 0.25 dwt. per ton, and the extraction 96 per cent.

In an article on the East Geduld in *Jour. S.A.I.E.*, January, 1933, H. R. Hill gives the total treatment cost as 30 pence or

60 cts. per ton and cyaniding cost as $9\frac{1}{2}$ pence or 19 cts. The flow-sheet in Fig. 114 is from the same article.

New State Areas Mill.—The New State Areas mill in the Witwatersrand area of South Africa treats 3900 tons per day. The information following has been supplied by F. Wartenweiler.

The ore is of the character usually found in this area and is described as quartz blanket reef. It assays 8.15 dwt. per ton in gold and 0.8 dwt. per ton in silver and contains some pyrite.

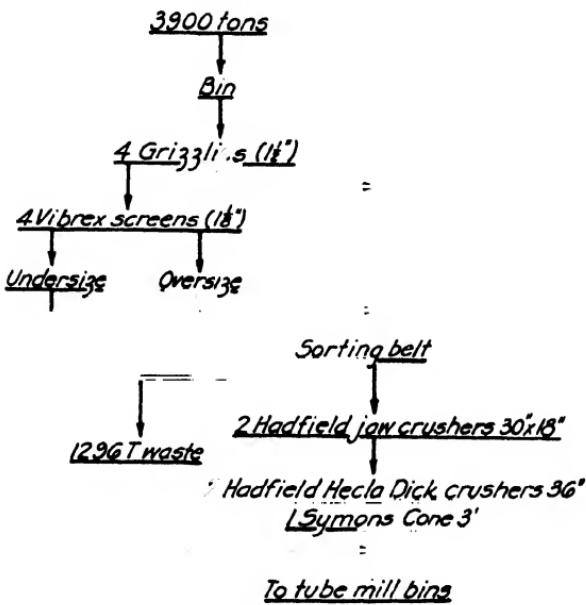


FIG. 115.—Flow-sheet of New State Areas crushing plant.

Crushing.—Figure 115 shows the flow-sheet of the crushing plant.

The grizzlies are set at $1\frac{1}{2}$ in.; the undersize goes to four Vibrex screens with $1\frac{1}{8}$ -in. square openings. The combined oversize from the screens and grizzlies goes to four sorting belts where about 1300 tons per day is discarded at a cost of 0.62 shilling (15 cts.) per ton feed. The screen undersize goes to mill feed, and the oversize is first crushed in two 30- by 18-in. Hadfield jaw crushers set at 2 in. and then in two 36-in. Hadfield Hecla discs and one 3-ft. Symons cone crusher set at $\frac{1}{4}$ and

$\frac{1}{16}$ in. respectively. The finished crusher product runs 13.7 per cent plus 1 in. and 52.1 per cent minus $\frac{1}{2}$ in.

Grinding.—Three-stage grinding as used in this mill is shown in Fig. 116. Primary grinding is done in nine 6.5- by 20-ft. tube mills each in closed circuit with a Dorr duplex classifier.

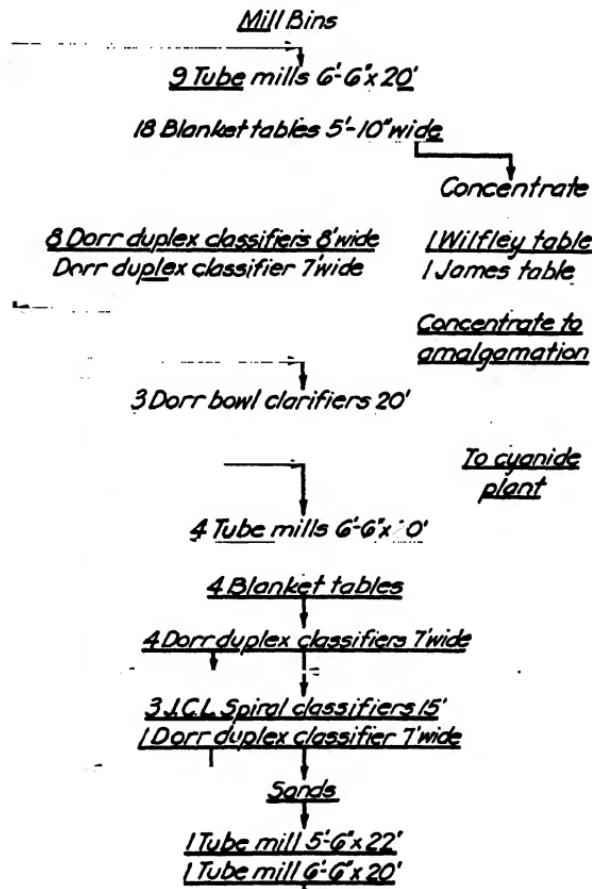


FIG. 116.—Flow-sheet of New State Areas grinding mill.

Eight classifiers are 8 ft. wide, and one is 7 ft. wide. The primary overflow sizing is 3.9 per cent plus 48 mesh; 24.3 per cent plus 100 mesh and 48.4 per cent minus 200 mesh. Tube-mill discharge flows over three corduroy tables, each 5 ft. 10 in. wide. The concentrate is cleaned on one Wilfley and one James table

and amalgamated in a barrel. Corduroy recovery is about 30 per cent.

The primary-classifier overflow goes to three Dorr bowl classifiers with 20-ft. diameter bowls. They overflow at 1.9 per cent on 100 mesh and 77.8 per cent through 200 mesh. About 2400 tons rake product goes to four secondary tube mills each

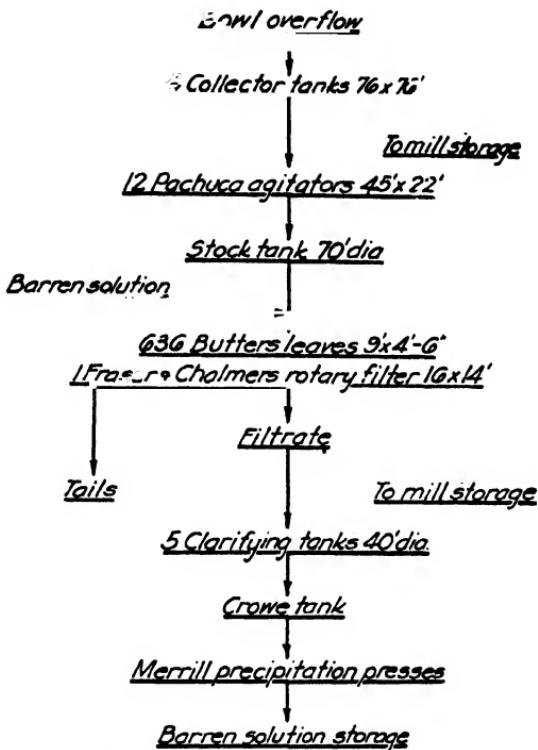


FIG. 117.—Flow-sheet of New State Areas cyanide plant.

in closed circuit with a Dorr duplex classifier 7 ft. wide. Each tube mill has one corduroy table similar to those in the primary stage. The secondary-classifier overflow goes to a tertiary grind in two tube mills, one 5.5 by 20 ft. and one 6.5 by 22 ft. in closed circuit with three 15-ft. J. C. L. spiral-pan classifiers and one 7-ft. wide Dorr duplex classifier.

The overflow of these classifiers returns to the intermediate bowl classifiers.

Slime Treatment.—Figure 117 shows the flow sheet of the slime-treatment section. The bowl overflow is thickened to 40 per cent solids on eight 76-ft. square settling tanks. The thickener underflow is given 15½ hr. agitation at 48 per cent solids in twelve 45- by 12-ft. Pachuca agitators.

Filtration is done on six hundred thirty-six 9- by 4.5-ft. Butters leaves and one 16-ft. by 14-ft. Fraser and Chalmers rotary filter. Part of the filtrate goes to precipitation, and the rest returns to the grinding solution. About 4500 tons of barren-solution wash is used per day.

Dissolved loss in the tailing is given at 0.02 dwt., and undissolved at 0.24 dwt. per ton.

Pregnant solution is clarified in five 40-ft. diameter tanks, and 4600 tons per day assaying 2.625 dwt. per ton is precipitated by the Merrill-Crowe process; zinc consumption is 0.073 lb. per ton ore.

Solution strength is 0.42 lb. KCN and 0.36 lb. CaO per ton, and the consumption is 0.51 lb. KCN and 2.12 lb. CaO per ton.

Lead nitrate is added to the clarifier tanks, ahead of the Crowe tank.

An extraction of 0.09 dwt. per ton is noted during the filtration step.

Refining.—The precipitate is acid treated with 45°Bé. sulphuric acid. The residue is water washed, filter pressed and then calcined before smelting. Sand, borax, fluorspar, soda ash and a small amount of manganese dioxide are used for smelting flux. The buttons are remelted into 1000-oz. bars. The amalgam is retorted and then remelted. Tails from the amalgam barrel are ground in closed circuit and cyanided.

Randfontein Estates.—The Randfontein mill in South Africa is the largest mill on the Witwatersrand and has a capacity of 12,000 tons ore per day. Concentration on corduroy tables, amalgamation and cyanidation treatment are all incorporated in the flow-sheets as shown in Figs. 118a and 118b.

The description of this mill and the accompanying flow-sheets and data are from a private communication to the author from F. Wartenweiler, consulting metallurgist of Johannesburg, dated May 21, 1935, supplemented by a paper of Wartenweiler's entitled "Development of Milling and Cyanidation on the Witwatersrand," in *Trans. 112, A. I. M. E.*, 1935. The ore is

described under "Witwatersrand," at the beginning of this section.

To quote from the foregoing A. I. M. E. paper, pages 760-761:

The Witwatersrand gold ore is named "basket" from the similarity of the oxidized outcrop ore to the Boer almond cake of that name. The interbedded ore body is commonly called "reef."

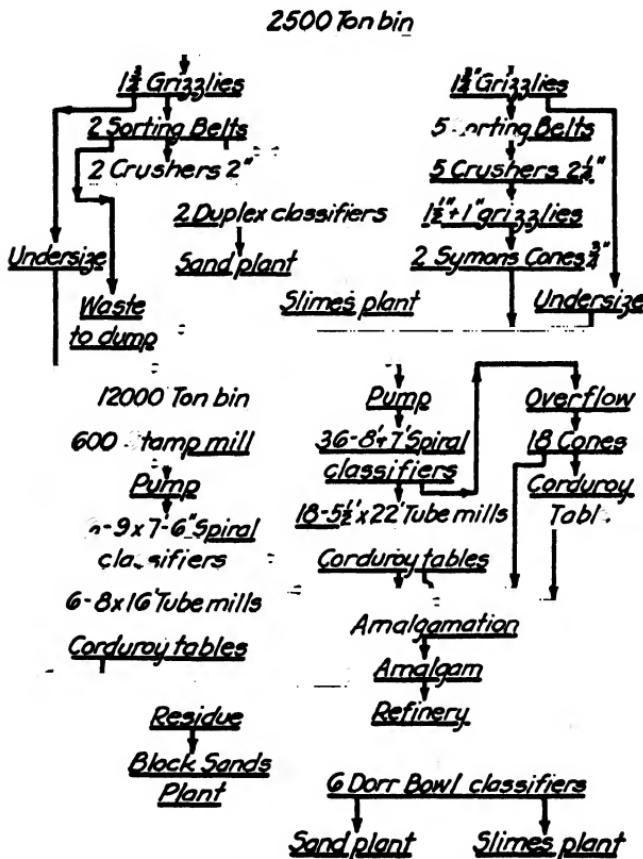


FIG. 118a.—Flow-sheet of crushing and milling plant, Randfontein Estates.

Milling grade at present price of gold (137 shillings per oz.) varies from 3.5 to 8.5 dwt., with one mine milling ore as high as 16 dwt. The gold-silver ratio is in the neighborhood of 9:1. Other characteristics are a high degree of abrasiveness; with depth (and stoping is proceeding as low as 7500 ft., vertical depth), it becomes denser and, owing to the high rock pressure, more brittle.

The gold in the ore may be divided into the portion that is in a free metallic state and the portion that is in the same state but closely associated with the pyrite, being found within and between crystal faces.

At Randfontein the ore is a conglomerate, containing the usual basket minerals common to the Witwatersrand. Main

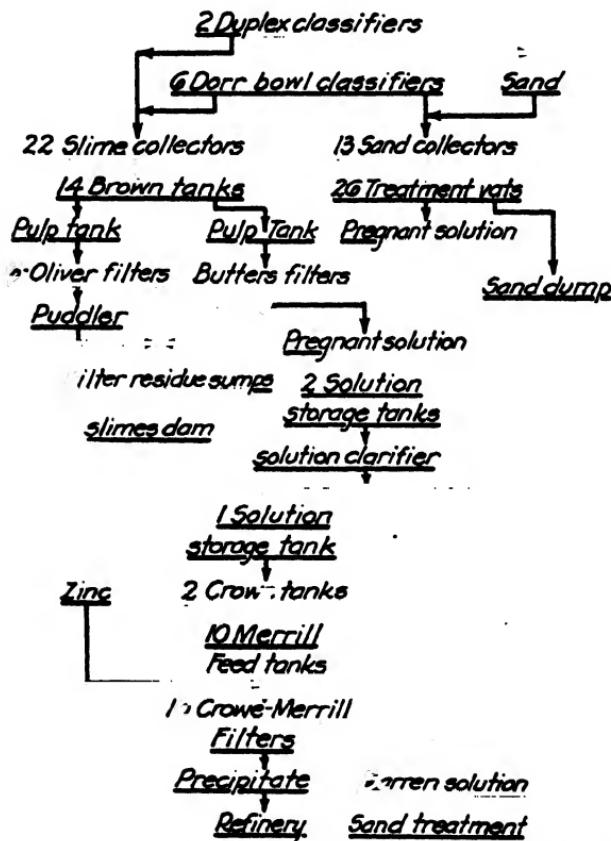


FIG. 118b.—Flow-sheet of cyanide mill, Randfontein Estates.

Reef ore averages 4.25 dwt. gold per ton; Kimberly-Elsburg, 3.29 dwt.; Bird Reef, 3.33 dwt.; Black Reef, 5.74 dwt., and Johnstone Reef, 3.79 dwt.

After passing over a $1\frac{3}{4}$ -in. grizzly, the oversize goes to sorting belts where 800 tons rock is rejected per day.

The sorting belts are sprayed with wash water, and material washed off the belt goes to two duplex Dorr classifiers. Two sorting belts feed two jaw crushers set at 2 in. The material through these two jaw crushers goes directly to a 12,000-ton bin. Five sorting belts feed five crushers set at $2\frac{1}{4}$ in. The product from the crushers goes to grizzlies set at $1\frac{1}{2}$ and 1 in. The undersize from these grizzlies joins the undersize from the $1\frac{3}{4}$ -in. grizzlies and goes to the 12,000-ton bin. The oversize from the $1\frac{1}{2}$ and 1-in. grizzly is fed to two Symons cone crushers set at $\frac{3}{4}$ in. The product from these crushers goes to the 12,000-ton bin.

Per Cent	Inches
2.9	+2
33.1	+1
15.8	+ $\frac{3}{4}$
18.0	+ $\frac{1}{2}$
30.2	- $\frac{1}{2}$
Total 100.0	

Crushed ore is fed to a 600-stamp mill, with a stamp weight of 1750 lb. and a capacity of 18.210 tons ore per 24 hr. Stamp product is pumped to six 9-ft. or 6 to 7 ft. 6 in. diameter primary spiral classifiers, operating at 13 r.p.m., in closed circuit with six 8- by 16-ft. primary tube mills, loaded with a combination of balls and pebbles. The critical mesh of separation is 100 (4711 tons per 24 hr.), and the tons of oversize or sand product are 6069 tons per 24 hr.

The discharge from each of the six tube mills passes over 4 ft. 4 in. by 6 ft. 3 in. corduroy concentration tables, set at an 18 per cent slope, on its way back to the classifiers in the closed grinding circuit. Twenty-eight tons of basket concentrate is recovered per 24 hr.

Primary classifier overflow is pumped to 36 secondary spiral classifiers, half of which number are 8 ft. in diameter and half 7 ft. All operate at 9 r.p.m. and overflow to 18-cone classifiers, 7 ft. 6 in. diameter by 6 ft. deep, which, in turn, overflow at 81 to 84 per cent minus 100 mesh. The classifier sand products or

oversize average 1968 tons per 24 hr. and are reground in 18 secondary tube mills, 5 ft. 6 in. in diameter by 22 ft. long, loaded with balls and pebbles. Cone-sand discharge is returned to the 8-ft. and 7-ft. spiral classifiers.

As in the case of the primary grinding circuit, the discharge of these secondary tube mills is passed over corduroy concentration tables before being repumped to the secondary spiral classifiers in the circuit. One set, 5 by 6 ft. and set at a 15 per cent slope, handles the tube-mill discharge, while another set, 5 ft. by 5 ft. 6 in. and set at a 14 per cent slope, handles the overflow of the final cone classifiers. Concentrate is dressed and amalgamated in barrels. Tailings from amalgamation are pumped to the sand plant for further treatment. Forty-eight per cent of the ore value is recovered by corduroy concentration in the primary and secondary grinding circuit.

The overflow from the cone classifiers in the secondary grinding circuit shows a sieve grading of 0.17 per cent plus 48 M, 14.19 per cent plus 100 M, 23.35 per cent plus 200 M and 61.57 per cent -200 M. This pulp is pumped to six Dorr bowl classifiers, 8 by 35 by 23 ft. in diameter, for separation into sand, 35.21 per cent of the total, and slime, 64.79 per cent.

Bowl classifier and duplex classifier sands are pumped to Butters-Mein distributors, serving 13 sand collectors, each 60 ft. in diameter by 12 ft. deep, which, in turn, discharge the collected sand to 26 leaching vats, 60 ft. in diameter by 10 ft. deep. When a leaching vat is full, 250 tons of 0.06 per cent KCN solution is pumped on over a period of 6 to 7 hr. A series of saturations are pumped on until the charge is ready for discharge. The ratio of solution to sand is 2.15 to 1. The residue assays 0.29 dwt. per ton.

All slime overflowing the bowl classifiers and the duplex classifiers is collected in twenty-two 70 ft. by 16 ft. by 7 ft. 6 in. cone-bottomed settlers. Supernatant water is decanted, and the slime transferred to 14 Pachuca or Brown agitators, 12 of which are 22 ft. in diameter by 45 ft. deep and two, 33 ft. in diameter by 45 ft. deep. Cyanide is added at this point.

In the agitators the solid content of the pulp is held at 50 per cent. Sixteen hundred cubic feet free air per minute at 60 to 70 lb. pressure is used in the treatment, and the total extraction is 93.68 per cent. After about 16 hr. treatment the pulp is

transferred to stock tanks, equipped with propeller-type agitators and air jets.

The filter plant consists of six Oliver vacuum filters, 14 ft. in diameter by 16 ft. face wide, and a Butters installation of 976 leaves, each 10 by 5 ft. The two dissimilar filter sections operate independently in parallel, each being served by a separate pulp-stock tank.

The six Olivers require about 3045 tons barren-solution wash per 24 hr. and deliver a 17 per cent moisture cake. The Butters filters use about 7514 tons barren solution per 24 hr. and yield a residue containing 28 per cent moisture. Tailings are pumped to waste.

Fifteen thousand tons pregnant solution from the sand and slime plants, at 1.30 dwt., is precipitated per 24 hr. with 0.05 lb. zinc dust per ton, to a value of 0.01 dwt., by the Merrill-Crowe system. Precipitate is hosed every 14 days from the Merrill leaves with KMnO_4 solution weak cyanide solution and is pumped to vats for treatment with 10 per cent H_2SO_4 . Pregnant solution is deaerated before precipitation, and the barren solution is aerated before re-use.

After acid treatment the precipitate is washed three times, pressed in Johnson pressure filters, calcined and charged into No. 100 crucibles with a flux of sand, borax and manganese dioxide. Crucibles, 30 per charge, remain in the furnace 3 to 4 hr. and, when poured, yield bullion bars containing up to 1000 oz. of gold. Bars are refined by a modified chlorine process, while slag is ground dry and run over riffles to recover possible gold for remelting. Final slag is assayed for gold, silver and platinum and shipped to a smelter for further treatment.

The data below reflect the efficiency of operation, the consumption of reagents and the solution strengths throughout the plant.

Recovery:

Mill-head value.....	3.76 dwt. per ton
Residue-slime treatment.....	0.112 dwt. per ton
Residue total (sand and slime).....	0.173 dwt. per ton
Dissolved value final tailing.....	0.02 dwt. per ton
Extraction total.....	95.40 per cent

Reagent consumptions:

Cyanide.....	0.189 lb. KCN per ton
Lime.....	2.188 lb. CaO per ton

NOTE.—Small amounts lead acetate or nitrate added to pregnant solution before clarification and precipitation.

Reagent strengths:

Cyanide in agitators.....	0.0120 per cent KCN
Lime in all circuits.....	0.0126 per cent CaO

Sub-Nigel.—This mine is the property of New Consolidated Goldfields and is located in the extreme southeastern area of the present large tonnage operations of the East Rand. Andrew King, consulting metallurgist, supplied the following data:

The plant is an all-slime plant of 2000 tons daily capacity, milling six days per week and cyaniding through seven days. The ore averages 15 dwt. gold and 1.5 dwt. silver per ton and about 2.7 per cent pyrite.

The ore is washed on three short, inclined belts after it has passed a $2\frac{3}{4}$ -in. opening grizzly and two double-deck screens. Two wash belts carry the minus $2\frac{3}{4}$ -in. plus $1\frac{3}{4}$ -in. sizes and discharge on to parallel picking belts, and a third belt carries the plus $2\frac{3}{4}$ -in. material and discharges on to a third picking belt. Waste and pebbles for grinding are sorted from the coarse sizes on the single belt and some waste and pebbles are sorted from the two belts carrying the finer material.

The largest material is minus 9-in. bar grizzly opening. Screen undersize minus $1\frac{3}{4}$ -in. plus $\frac{3}{4}$ -in. goes over a washing grizzly to crusher feed bins ahead of four No. 7 Newhouse crushers. Minus $\frac{3}{4}$ -in. material goes to mill feed bins where it is joined by the Newhouse crusher product.

Washings go to a Dorr F classifier with the classifier rake product going to mill feed bins and the classifier overflow to a 50-ft. Dorr thickener, which overflows clear water to storage for reuse. The thickener underflow solids are pumped to the secondary grinding circuit.

Fine grinding is done in ball-and-tube mills in two stages. Primary reduction to 50 to 55 per cent — 200 mesh in four ball mills 6 ft. 6 in. by 9 ft. and four tube mills 6 ft. 6 in. by 20 ft. in closed circuit with eight Dorr F single-stage classifiers. Secondary reduction is to 85 to 90 per cent — 200 mesh in five tube mills 6 by 20 ft. in closed circuit with six type F Dorr bowl classifiers.

With the shortage of pebbles on the East Rand, Sub-Nigel in late 1935 and early 1936 installed four primary 6 ft. 6 in. by 9 ft. ball mills and will thereby conserve pebbles for the secondary

grinding operation. These units replace four primary tube mills 6 ft. 6 in. by 20 ft. Composite loads were tried in the primary tube mills and found unsuitable on account of poor ore pebbles. All-pebble loads are used in the four primary tube mills and in the secondary mills.

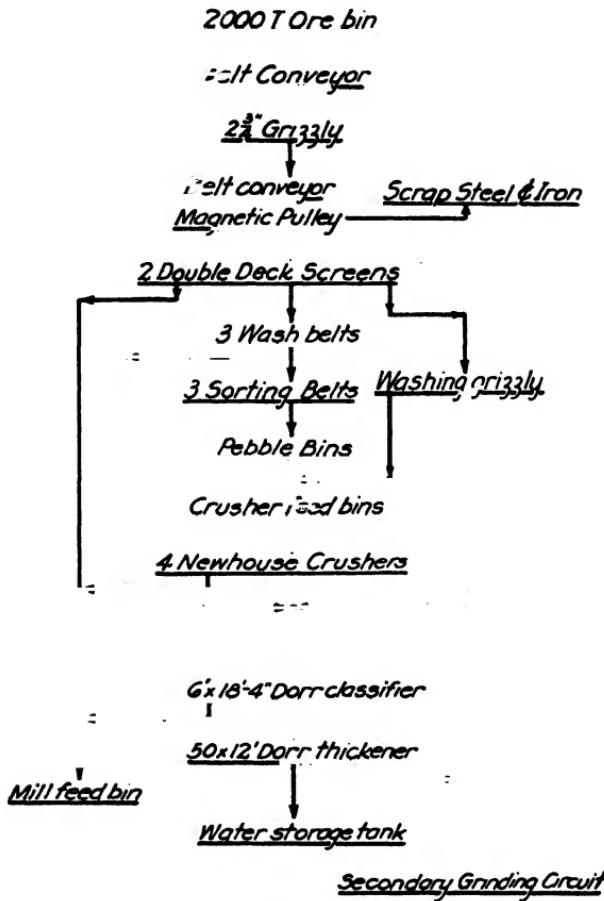


FIG. 119a.—Flow-sheet, crushing, washing and sorting, Sub-Nigel.

Cyanide strength in the grinding circuit is maintained at approximately 0.002 per cent KCN. Corduroy tables in the primary circuit recover 55 to 60 per cent of the total gold. Corduroy concentrate is put over a Wilfley table. The table concentrate is amalgamated. Table tails are returned to the grinding circuit.

The pulp from the secondary circuit overflows the bowl classifiers and gravitates to eight thickeners, three 50-ft.-diameter Dorr connected-type tray thickeners, and five 50-ft.-diameter Dorr unit-type thickeners. The underflow from each tray thickener is controlled by a Dorreco quadruplex suction pump, and each unit thickener by a Dorreco duplex suction pump.

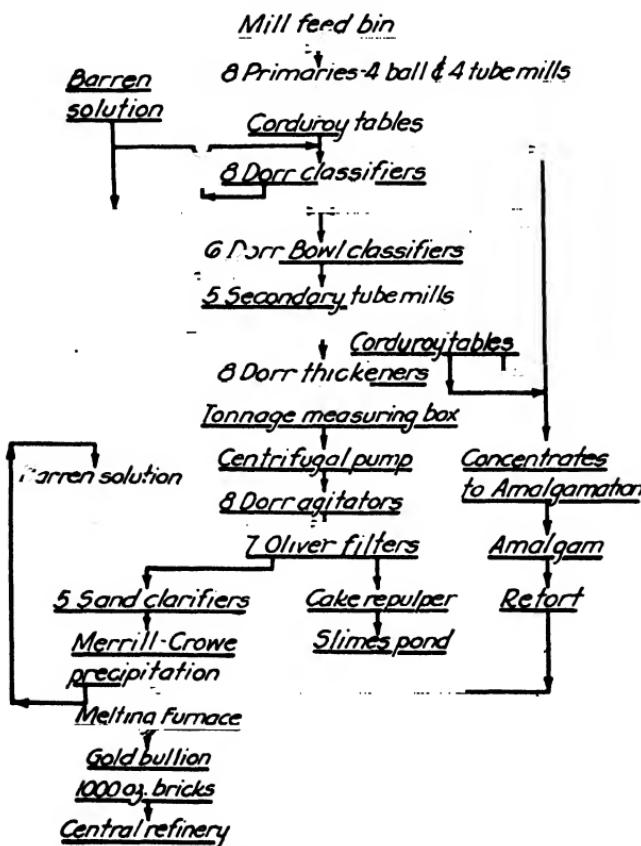


FIG. 119b.—Flow-sheet, milling and treatment, Sub-Nigel.

All thickeners are equipped with Bowley drives. Sufficient pulp is accumulated in the thickeners over the six-day milling week to keep the cyanide plant in operation during the seven days.

Suction diaphragm-pump discharges gravitate to a tonnage-measuring box where constant check on mill tonnage is made.

From this box the pulp gravitates to the suction of a 10-in. Morris pump where strong cyanide is added and the pulp, averaging about 1.65 sp. gr., is pumped to a two-way distributor ahead of eight agitators in parallel rows of four each. Pulp enters the agitators at 1.35 to 1.38 sp. gr.

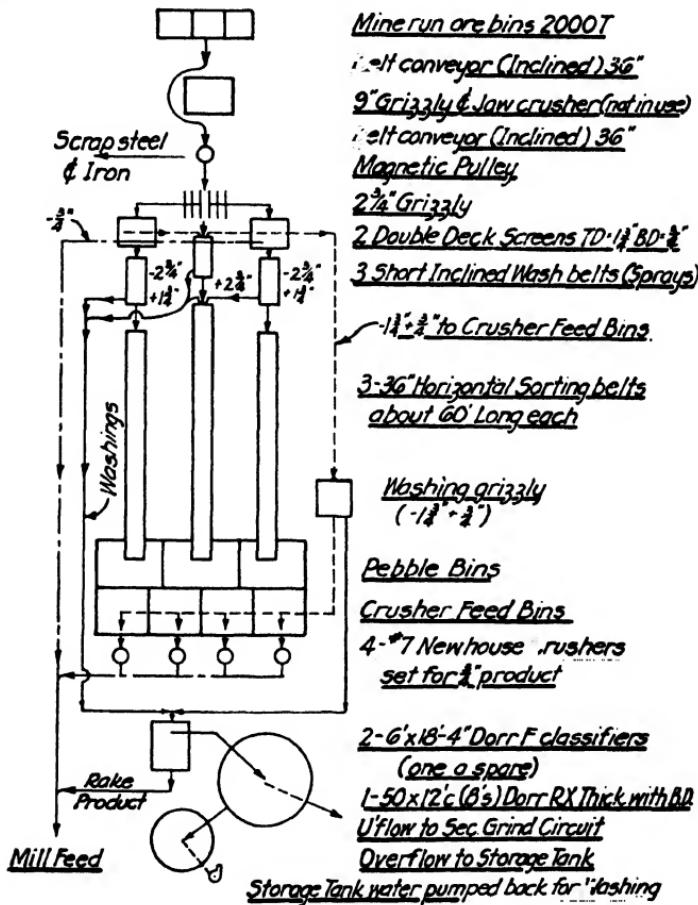


FIG. 119c.—Flow-sheet of Sub-Nigel crusher and sorting stations.

The heads at the distributor average 6 dwt. gold and the solution carries 0.025 per cent lime and 0.028 per cent KCN.

The agitators are 50 ft. diameter by 12 ft. depth at the outside and 16 ft. depth at the center. The period of agitation is approximately 42 hours.

The last two agitators are interconnected and discharge to seven Oliver filters, three 14 by 16 ft. and four 11 ft. 6 in. by 14 ft., where, after dewatering and a barren-solution wash, the cake is discharged at about 27 per cent moisture.

The filter cake is repulped and gravitates to a disposal area from which clear barren solution is pumped back to mill solution tanks. The filtrate is clarified in five sand clarifiers, two of 40-ft. diameter and three of 30-ft. diameter, with sand beds 18 to 24 in. deep. The sand beds are made up from rake product of the primary classifiers.

After clarification, the rich solution goes through Merrill-Crowe precipitation system. Zinc dust consumption is 0.1 lb. per ton of ore. Precipitation ratio 1.7 to 1. Rich solution is $3\frac{1}{2}$ to 4 dwt., barren solution 0.02. Over-all recovery is 98.3 per cent. Lead acetate is added prior to filtration.

The gold mud from the Merrill-Crowe precipitation system and the amalgam are furnace down to gold bricks of 1000-oz. each, which are shipped to the Central Rand Refinery.

Wanderers—Selukwe, Southern Rhodesia.—The ore from this mine is from a crushed zone of shale between banded iron-stone and conglomerate mineralized by quartz carrying auriferous pyrite.

Mill heads are 0.225 oz. gold and 0.010 oz. silver per ton. There is a small quantity of copper. Crushing is in two stages through Hadfield 30-in. by 15-in. jaw crushers and through 36-in. Helga disc secondary crushers to $1\frac{1}{2}$ in.

Grinding is in two stages with one 6- by 20-ft. and three 5 ft. 6 in. by 22 ft. primary tube mills in closed circuit with three 7- by 6-ft. cones and one secondary tube mill 5 ft. 6 in by 2 ft. in closed circuit with a Dorr bowl classifier with 15-ft.-diameter bowl.

Concentrates from eight Record vanners in the primary circuit are reground in a concentrate tube mill 5 ft. 6 in. by 18 ft. with blanket tables in the circuit. Concentrates from the blanket tables go to barrel amalgamation.

Pulp from the bowl classifier is thickened in three main Dorr thickeners 30 by 10 ft. with underflow to three main Dorr agitators, two 40 by 10 ft. and one 50 by 12 ft. Here it is joined by the pulp overflowing the cones in the concentrate reground circuit which has been thickened in a 30- by 10-ft. Dorr thickener.

and passed through two Dorr concentrate agitators each 16 by 8 ft.

Agitation is for 32 hours at approximately 40 per cent solids and pulp from agitation is filtered direct on two Oliver filters,

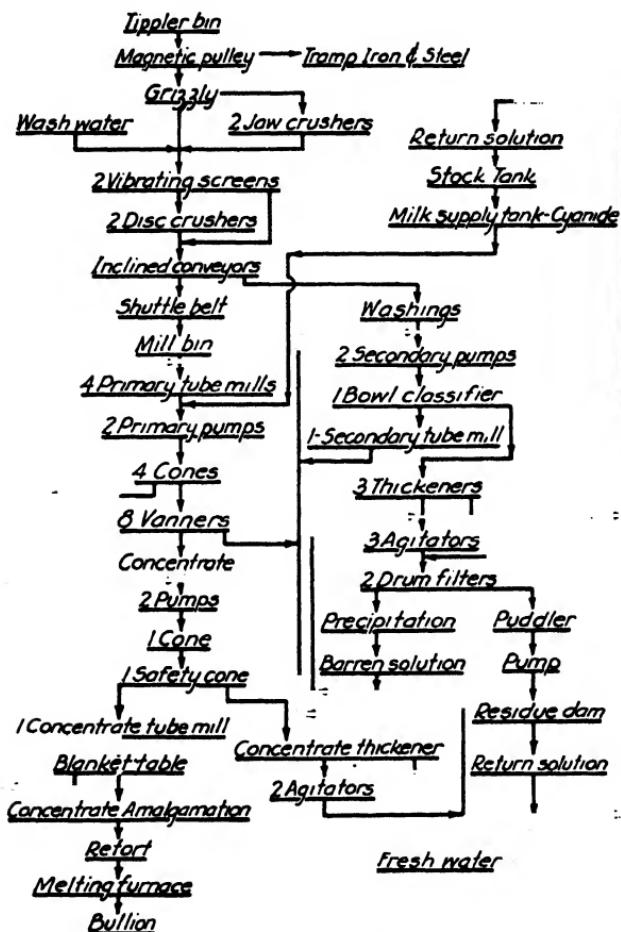


FIG. 120.—Flow-sheet, Wanderers' Consolidated Gold Mines, Ltd.

one 11 ft. 6 in. by 16 ft. and one 14 by 16 ft., to a cake moisture of 18 per cent.

Barren-solution wash on the filters is followed by a wash with combined fresh water and return solution from the slime tank. Filter cake is repulped and pumped to the residue dam.

Rich solution, value 1.0096 dwt. per ton, from the filter station goes to a Merrill-Crowe system. One thousand six hundred and eight tons per 24 hr. are precipitated to barren solution of 0.0085 dwt. per ton. Zinc dust consumption is 0.097 lb. per ton of ore.

Grind is to 85 per cent -200 mesh, with a concentrate regrind to 98.35 per cent -200. The ore is ground in cyanide solution. Cyanide strength at the first agitator is 0.03 to 0.04 per cent KCN. Consumption of cyanide per ton of ore is 0.567 lb.

Protective alkalinity is carried at 0.280 lb. CaO per ton of solution with a consumption of 2.955 lb. CaO per ton of ore. Lead acetate or nitrate is added at No. 1 concentrate agitator, 2 lb. per shift; No. 1 agitator 2 lb. per shift; solution clarifier 1 lb. per shift. Dissolved value loss in the final solution is 0.022 dwt. per ton and undissolved value is 0.429 dwt. per ton.

Recent information from Andrew King, consulting metallurgist, is to the effect that Wanderers' practice has just recently been changed and now includes flotation with flotation tails to waste and cyanidation of reground concentrate. More detailed information is not yet available.

West Rand Consolidated.--C. S. McLean, general manager, furnished the following flow-sheet and information:

The capacity of the mill is 120,000 tons per month (4000 tons per 24 hr.); mill heads 4.7 dwt. gold, 0.45 dwt. silver, 3.1 per cent pyrite.

Primary crushing is done in a Hadfield jaw crusher 30 by 18 in. set at 8 in. and operating 24 hr. a day. Primary screening is in trommels with $2\frac{1}{2}$ -in. holes. Pebble sizing is in trommels. Secondary crushing is done in Newhouse gyratory crushers set at 1 in. Sorted rejects are 600 tons primary, 20 tons secondary.

Crushing plant screening is on secondary Robins Gyrex screens with $1\frac{1}{2}$ -in. openings and on tertiary Vibrex and Gyrex screens with $\frac{3}{4}$ -in. openings. Natural fines of $-3\frac{1}{4}$ in. and undersize $-3\frac{1}{4}$ in. from secondary crushers go to the new tube mill plant. All $+3\frac{1}{4}$ -in. material goes to 100 stamp mills that serve as tertiary crushers, and thence to the old tube mill plant. The dewatering screens are Hum-mers with No. 10 screen cloth. Mill storage bins have a capacity of 5000 tons. There are seventeen primary tube mills—eleven 6 ft. 6 in. by 20 ft., five 5 ft. by 16 ft., one 6 ft. by 16 ft.

In the new plant there are eleven Dorr straight classifiers—eight 8 ft. by 18 ft. 6 in. and three 8 ft. by 13 ft. In the old plant are six Dorr straight classifiers 4 ft. 6 in. by 18 ft. 6 in. Primary separation is at 48 mesh.

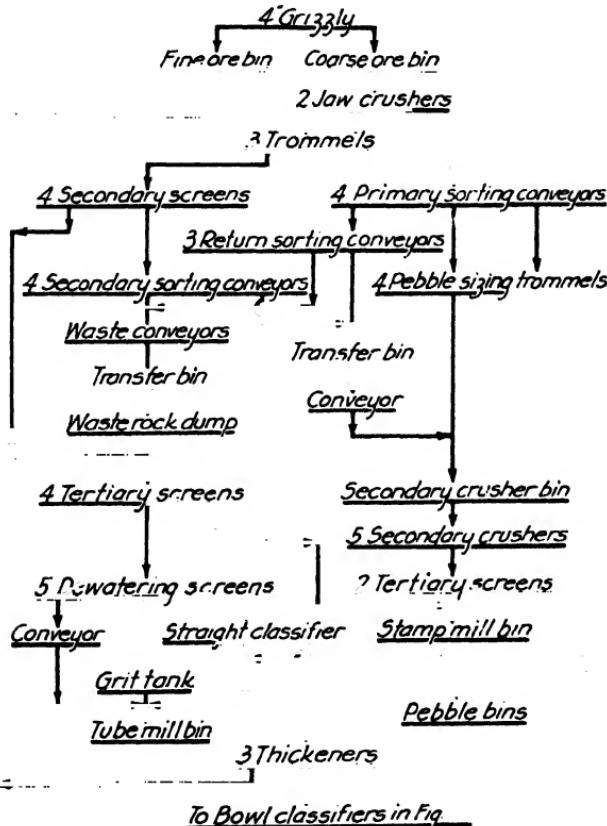


FIG. 121a.—West Rand Consolidated, general flow-sheet (first section).

One 6-ft. diameter Hardy Smith jig is inserted in each primary tube mill receiving most of the mill discharge and making a concentrate at 13 per cent pyrite and 15 dwt. gold. Total tons of concentrate equal 610.

There are four Dorr bowl classifiers in the secondary circuit, each 6 ft. by 26 ft. 8 in. by 16 ft. diameter bowl, making separation at 100 mesh.

There are four 6-ft. Hardy Smith jigs receiving the rake product from the four bowl classifiers, making a total of 80 tons of concentrate running 13 per cent and 15 dwt. gold.

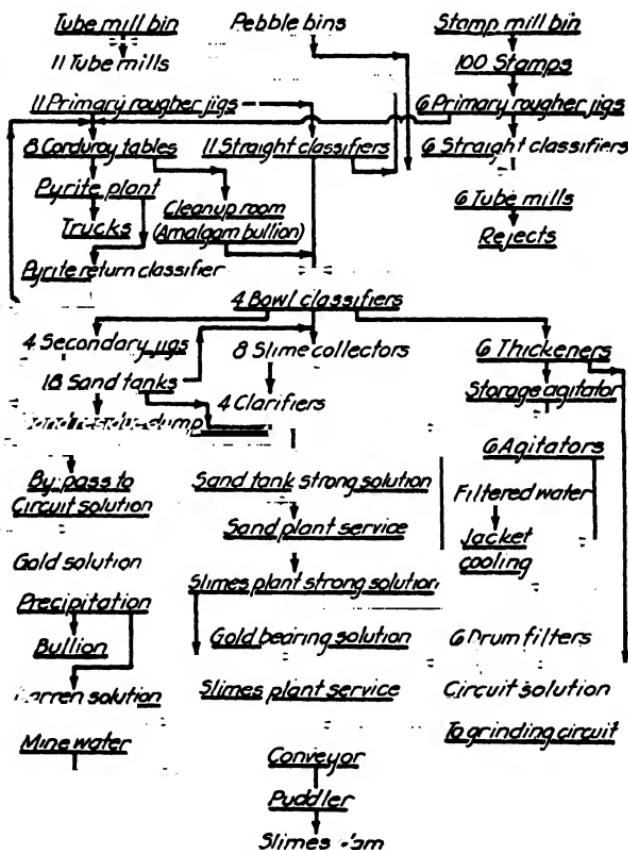


FIG. 121b.—West Rand Consolidated, general flow-sheet. (Continued.)

Sieve grading of final (Dorr bowl) classifiers is as follows:

	+48	+100	+200	-200
Sand.....	5.2	54.1	35.3	5.4
Slime.....	...	2.3	21.1	76.6

The sands go to six sand tanks 42 ft. by 8 ft. and twelve sand tanks 40 ft. by 12 ft.

Solutions and washes are as follows:

	Key	Alk.	Dwt.	Tons solution per tank
Strong NaCy solution used.....	0.055	0.023	2.0	320
Medium NaCy solution used.....	0.009	0.024	2.0	340
Precipitated NaCy solution used.....	0.014	0.025	0.02	450
Water wash used.....	30
Total.....	1140
Sand residue average value 0.25 dwt.				
Last drains average value 0.10 dwt.				
Ratio.....				2.51 1

The slimes go to eight standard slime collectors 50 ft. by 12 ft. and six Dorr thickeners 50 by 12 ft. The pyrite concentrate goes to a separate treating plant (see Fig. 122).

Thickened slimes are batch agitated in six Brown tanks without center column—diameter 22 ft., depth of cone 18 ft. 3 in., depth of cylinder 35 ft. Specific gravity of pulp is 1.37, per cent solids 42.9, total hours contact 10, cubic feet of free air per minute total 2000, air pressure 35 lb.

Agitator discharge is filtered direct on six Oliver filters, 14 ft. diameter by 16 ft. long. Cake moisture is 26 per cent and 0.692 ton of barren-solution wash per ton of slime. Barren wash water is 0.1 ton per ton of slime.

Loss of dissolved value in final tailing—slimes only—is as follows:

Gold.....	140 shillings per oz.	Gold.....	0.189 shilling per ton
Silver.....	2 shillings per oz.	Silver.....	0.0003 shilling per ton

Total..... 0.1893 shilling per ton

Loss of undissolved value in final tailing—slimes only—is as follows:

Gold.....	0.637 shilling per ton
Silver.....	0.0009 shilling per ton

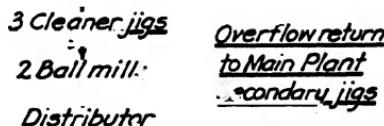
Total..... 0.6379 shilling per ton

Over-all extraction for the whole plant—sand, slime, pyrite—is 96 per cent. Value of pregnant solution—3.04 dwt. at 7 shillings per dwt.—is 21.28 shillings; value of barren solution at above metal value, 0.14 shilling. Tons precipitated per 24 hours, 4000; zinc consumption, 0.049 lb. per ton of ore.

The Merrill-Crowe system is used. Gold slime is washed from the Merrill press leaves and transferred to an acid tub and

Pyrte Plant

1. Watering spitzkastens



12 Concentrating tables

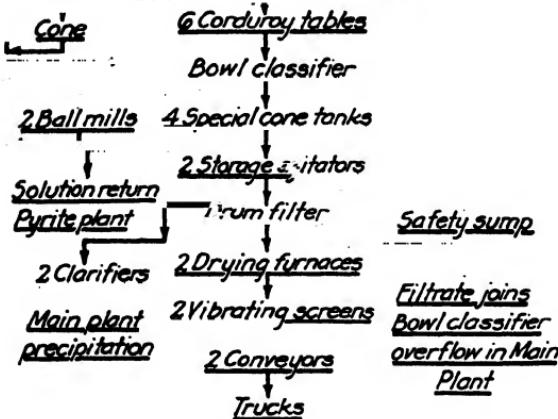


FIG. 122.—West Rand Consolidated, flow-sheet of pyrite mill.

sulphuric acid is added. After decanting and water washing, gold slime is put through a pressure filter. The filter cake is calcined, then smelted with sand, borax and MnO_2 in a reverberatory furnace. Buttons are remelted with borax. In the clean-up mill mercury is retorted from the amalgam and the gold smelted with borax.

Concentrates from rougher jigs in the tube mill circuit go to cleaner jigs. Concentrates from cleaner jigs go to James tables,

from which the gold streak goes to amalgamation and concentrate via corduroy tables to the ball mill regrind. Corduroy concentrate also goes to amalgamation. Reground pyrite (85 per cent FeS_2) is treated in Crosse tanks. Residue at 0.9 dwt. is settled, decanted, water washed and filtered on an Oliver filter. The filter cake is dried, shipped and sold.

Cyanide is added at the tube mills in the new plant and at the stamp mills in the old plant. The strength of cyanide solution entering agitators is 0.28 lb. KCN per ton solution. Consumption is 0.11 lb. KCN per ton of ore. The over-all consumption of cyanide in the whole plant is 0.214 lb. per ton ore.

Protective alkalinity is 0.28 lb. CaO per ton solution entering the Brown agitators. The consumption of lime in these agitators is 0.13 lb. per ton ore. The over-all consumption of lime in the whole plant is equal to 1.694 lb. per ton ore. Lime used is 90 per cent CaO.

Other chemicals consumed are lead nitrate, 0.012 lb. per ton of ore, added at the Merrill presses and also in the pyrite plant at Crosse tanks and to the sand charges. Potassium permanganate, 0.012 lb. per ton of ore, is added at the crusher station and at the Dorr bowl classifiers. In treating gold slime 0.049 lb. sulphuric acid per ton ore is used.

Dissolution in the treatment plant shows:-

Slime entering thickeners.....	1.75 dwt. (washed)
Slime leaving thickeners.....	1.26 dwt. (washed)
Slime entering filters.....	0.182 dwt. (washed)
Slime leaving filters.....	0.091 dwt. (washed)

West Springs Mill.—The West Springs mill in South Africa receives about 4000 tons per day of which approximately 340 tons is discarded by sorting.

Crushing is done by two 6-N Gates gyratory crushers and one 7-ft. Newhouse gyratory crusher. These produce a feed which is about 80 per cent minus 1 in.

Primary grinding is done in nine tube mills, 9 ft. 9 in. by 20 ft., in closed circuit with 8-ft. by 18-ft. 4-in. Dorr duplex classifiers. A circulating load of about 1.2 to 1 is carried. The plus $\frac{1}{2}$ -in. product screened from the tube-mill discharge is reground in three Hecla disc crushers. The primary classifiers overflow at 24.2 per cent plus 48 mesh and 35.7 per cent minus 200 mesh.

This overflow goes to two Dorr bowl classifiers, 8 by 30 with 20-ft. diameter bowls, in closed circuit with four 6.5- by 20-ft. tube mills. These classifiers rake about 2600 tons sand per day, and since their installation improved operation through selective grinding of sulphides has been noted.

The secondary mills are loaded with a mixture of 2-in. cast-iron balls and plus 1½-in. reject pebbles from the primary tube mills. Bowl overflow is 9.5 per cent plus 100 mesh and 65.8 per cent minus 200 mesh.

The pulp is thickened in seven 70- by 12-ft. plain tanks and is discharged at 30 per cent solids.

Agitation after thickening is done at about 52 per cent solids in eleven 45- by 15-ft. Pachuca agitators. Total period of agitation is 8½ hr. From 850 to 900 cu. ft. free air per min. is supplied to the agitators at 35 lb. per sq. in. pressure. Filtration is on 490 Butters filter leaves, 4050 tons per day barren-solution wash being used.

Tailings losses are given as 1.56 pence (3.12 cts.) soluble and 1.109 shillings (26.3 cts.) undissolved per ton.

Zinc dust is used to precipitate 3800 tons of solution per day assaying 3.5 dwt. per ton.

Solution strengths are 0.298 lb. KCN and 0.36 lb. CaO per ton solution, and the consumption amounts to 0.28 lb. KCN and 1.75 lb. CaO; in addition, lead nitrate is added in the following amounts: 10 lb. per shift to the tube mill; 5 lb. to each Pachuca agitator and 2 lb. ahead of precipitation.

COMPLEX ORES

This section of the chapter on ore treatment covers the treatment of complex ores, such as those containing arsenic, copper, manganese, pyrrhotite and tellurides, with illustrations from actual practice.

THE USE OF MERCURY IN THE CYANIDE PROCESS

South American Development Company.—The following notes are taken from a private communication from Paul C. Schraps of Oakland, Calif., describing his patented cyanide-mercury process, as developed by him for the South American Development Company at Porto Velo, Ecuador.

The ores of these mines are roughly classified as (1) calcitic quartz ores and (2) complex ores. The process in question was developed to treat the complex ores, containing up to 30 per cent of sulphides consisting of galenite, sphalerite, pyrite, chalcopyrite, pyrrhotite and tetrahedrite; also quartz, calcite and native copper.

The relatively simple calcitic quartz ores were amenable to straight cyanidation, which yielded a 95 per cent recovery of the gold. The complex ores, however, under test plant conditions, did not react well to any type of cyanide treatment, the maximum recovery obtained being about 85 per cent, even when grinding to 200 mesh. Flotation in combination with cyanidation did not increase extraction appreciably and direct shipment of concentrates was prohibitive in cost in view of the isolated location of the mine.

Stamping, grinding and classifying to 80 per cent -200 mesh in cyanide solution, followed by thickening, result in a 60 per cent dissolution of gold.

Dilution of thickener underflow with cyanide solution to a 3 to 1 ratio of solution to solids by weight is followed by treatment in 4- by 10-ft. cylinders, revolved at 17 r.p.m. and containing 50 lb. or less of metallic mercury, with the discharged pulp passed over tables to catch entrained mercury.

There is thickening of the tabled pulp, followed by agitation for the further recovery of silver and finally counter-current decantation in a series of thickeners.

The capacity of the 4- by 10-ft. revolving cylinders was 50 dry tons per 24 hr. The final extraction of gold was as high as 97 per cent. Amalgamation, strictly speaking, was not involved, since the same charge of mercury was used in the cylinders for 25 days of continuous operation, whereupon it was strained and re-used with a yield of only a few ounces of amalgam.

According to the inventor, contact of the gold with the metallic mercury accelerates the dissolving action and at the same time puts the refractory gold in suitable condition for dissolution in cyanide solution. The explanation may lie in the gold's being coated with some substance that prevents the solution from coming in contact with it without the mediumship of metallic mercury. The soluble mercury salts gave negative results, both in the laboratory and in the test plant.

This process was developed in 1928 for the treatment of the complex ores in a separate circuit, which gave quite satisfactory results with upwards of 10 per cent increased extraction, as compared with straight cyanidation. Cyanide consumption was quite high, but the treatment with mercury, as well as cyanide regeneration, reduced this cost to a reasonable figure.

In the present flow-sheet the complex ores are mixed with the quartz ore and the same treatment is given the mixed ore through the grinding circuit. Following grinding the ore is classified and the fine sands concentrated on tables, after which the concentrate is treated in the mercury cylinders. Tailing from the mercury treatment is agitated in a somewhat higher strength cyanide solution than is used in the main mill circuit. After agitation the agitator discharge goes into the general mill circuit for continuous countercurrent washing.

During 1935, 35 per cent of the mill heads consisted of complex ore; mercury consumption averaged 0.015 lb. per ton of ore at a cost of 0.012 ct. per ton.

ARSENOPYRITE IN ORE

The profitable treatment of 1400 tons per day of ore containing \$9 in gold, 50 per cent silica, $2\frac{1}{2}$ per cent arsenopyrite and 3 per cent iron pyrite at Wiluna, Western Australia, is noteworthy. After flotation the tailing is discarded, and the concentrate treated by roasting, grinding and cyanidation. Milling and treatment costs are low, at around 7 shillings (\$1.68) per ton of ore, and recovery exceeds 90 per cent. The arsenic is saved and refined. Various treatments have been given the arsenical ores from the Cam and Motor, Rhodesia (also antimony bearing); Hedley, British Columbia; Jardine, Mont.; Keystone, S.D.; La Belliere, France, and Passagem, Brazil. All made a concentrate which was treated raw, shipped to the smelter or roasted and cyanided.

Cam and Motor.—The ore of the Cam and Motor mine in Rhodesia presents many metallurgical problems. It contains quartz with silicate of alumina, carbonate of lime and magnesium and iron pyrite, arsenopyrite and stibnite intimately mixed. The antimony, magnesia and arsenic make the ore difficult to cyanide. Prior to 1919, treatment following Kalgoorlie practice and consisting of all-dry crushing, roasting, sliming and cyaniding

gave an extraction of 74 per cent. Since then, several changes have been made in the treatment, the present method recovering 90 per cent of the gold, according to Digby Burnett, general manager and consulting engineer. Briefly the flow-sheet is as follows: 56 Nissen stamps crush the ore after which 17 Record vanners carry out primary concentration. Eight Dorr classifiers handle the vanner tailings and pass the coarse material to five tube mills. After regrinding, 19 Record vanners perform the secondary concentration, and all concentrates are reconcentrated on three Record vanners. The concentrates then pass to amalgamating barrels and copper plates and eventually to the roasting furnaces. The classifier overflow passes over 2600 sq. ft. of blankets, thence to three cones and nine thickeners the underflow from which is pumped to 28 James slime tables and six James slime-reconcentrating tables. The concentrate from all tables is roasted in five Edwards duplex roasters. Weight loss during roasting is 12 per cent, and value loss 4 per cent. Concentrates assay 22.25 dwt. per ton after roasting and are given a 14-day treatment with a solution containing 3.0 lb. per ton KCN and 1 lb. per ton CaO. All other pulp, after concentration, is wasted.

An interesting feature as described by Digby Burnett in *M.M.*, October, 1935, is the weathering and retreatment of the cyanided concentrates. They are left on the dump for 6 months and then reground and given a second cyanide treatment in slime tanks and sand vats. An additional 2 per cent extraction is credited to this operation.

PYRITE IN ORE

The Canyon Corporation.—The so-called blue ores of South Dakota have long represented a difficult treatment problem. In 1935, the Canyon Corporation erected a 125-ton roasting and cyanide plant on the Maitland mine near Deadwood and with only a few months of operation report quite satisfactory results.

The Maitland ore bodies occur in fractures in dolomite and consist of hard, glassy quartz containing extremely finely disseminated pyrite with which the gold is associated. The pyrite grains vary from 2 to 3 mm in diameter to as low as 0.001 mm and probably finer.

Carefully controlled roasting is necessary in order to prepare the ore for satisfactory cyanide treatment. The theory on which the roasting practice is based is slowly to heat in an oxidizing atmosphere to burn off the first sulphur atom and continue at this temperature to form iron sulphate. After the sulphate is formed, the temperature is increased above the

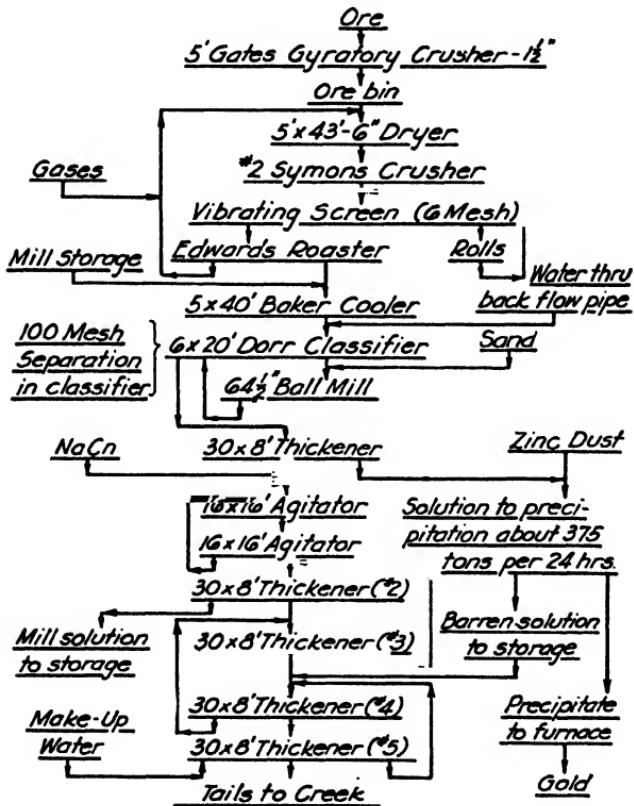


FIG. 123.—Flow-sheet, Maitland mine, Canyon Corporation.

decomposition point of the sulphate until the iron is converted to Fe_2O_3 . By experiment, a temperature curve has been developed to control the roasting.

Figure 123 shows the general flow-sheet of the plant. Following preliminary crushing to $1\frac{1}{2}$ in., the ore is dried in a cylindrical dryer and further crushed to 6 mesh after which it is roasted in a 54-spindle, oil-fired Edwards roaster, cooled and immediately

added to cyanide solution in the fine-grinding section where it is ground in closed circuit through 100 mesh, with 70 per cent minus 200 mesh agitated with a change of solution between the second and third agitator and washed in a standard c.e.d. plant.

The cyanide consumption including mechanical loss in the tailing runs from 1.0 to 1.1 lb. per ton ore, and the lime consumption from about 10 to 12 lb. per ton. Extraction runs close to 85 per cent on a head slightly less than $\frac{1}{2}$ oz. gold per ton.

The roasted ore settles readily to from 58 to 60 per cent solids in the thickeners. Extraction takes place slowly throughout the agitator series probably owing to extremely fine distribution of values in the ore and to incomplete oxidation of all portions of the sulphides.

The cost of preliminary crushing and drying runs slightly over 50 cts. per ton and of roasting from 55 to 60 cts. per ton.

PYRRHOTITE IN ORE

The methods used at the Homestake and at Morro Velho represent the special treatment made necessary by pyrrhotite ores. In the ores of both mines, the gold occurs free and associated with pyrrhotite and pyrite. Arsenopyrite is present also. Many other mines produce ore in which pyrrhotite is a constituent. The ores at Porecupine, in the Kolar district and on the Rand show pyrrhotite, but its effect is not so pronounced as at the mines already mentioned.

The notes following are abstracted from articles by A. J. Clark, metallurgist for the Homestake Mining Company, in *E. and M.J.*, Oct. 12, 1931; and by J. H. French and Harold Jones of the St. John del Rey Mining Company in *Bul. I.M. and M.*, December, 1932, and February and March, 1933.

Homestake Treatment.—Among the minerals containing iron in the ferrous state are pyrrhotite, chlorite, cummingtonite (an iron-magnesium amphibole) and a dolomitic carbonate. Some of these carry as much as 30 per cent iron. Arsenopyrite is also a constituent. These minerals oxidize rapidly, perhaps it would be more precise to say steadily, as the action continues seemingly throughout the treatment cycle. In oxidizing they generate cyanicides and withdraw oxygen from the working solutions. Cyanidation must be applied immediately after comminution.

tion, and oxygen must be supplied from time to time to maintain dissolution of the gold. Crushing in water, rather than in cyanide solution, is desirable.

The fundamental requirements for cyanidation of Homestake ore, established in the early years of this century, are the same today, according to A. J. Clark, metallurgist, in *E. and M.J.*, Oct. 12, 1931. These fundamentals are: (1) Amalgamation is suitable for saving much of the gold; (2) ferrous compounds in many of the gangue minerals necessitate careful chemical and mechanical preparation of the pulp for cyanidation; and (3) the iron sulphides and arsenopyrite yield their gold readily and at fairly coarse sizes; the nonsulphide minerals, even when finely ground, yield gold reluctantly. As details of leaching and other features of treatment are given in other sections of this book, only a generalized description follows here.

Crushing.—Mine-run size ore is first crushed underground in one shaft and on the surface at another shaft. Second-stage crushers at the surface of both shafts produce 1½- to 2-in. ore for the stamps. Crushing costs 3.8 cts. per ton.

Only the South mill of 180 stamps is now at work. It crushes 3800 tons daily. The stamps weigh 1570 lb. each when new, are arranged in two rows back to back with ore bin between and each group of five is driven separately. They crush up to 21 tons each per day.

Amalgamation.—This is done now in Clark-Todd amalgamators, 10 of which are in the rod-mill circuits. Mercury is fed to these mills, and about 63 per cent of the gold is recovered at this stage. The remaining free gold is fine enough for dissolution in cyanide.

Grinding and Classification.—The discharge from the stamps goes to dewatering cones, the underflow of which is fed to Allis-Chalmers and Marcy rod mills, which are in closed circuit with Dorr classifiers.

Overflow from the rod-mill section amounting to 1000 tons per day is dewatered in a second series of cones. The underflow from the cones is ground in Allis-Chalmers tube mills in closed circuit with Dorr classifiers. The pulp advances to gravity and hydraulic cones, the underflow of which is sent to sand-leaching vats. Overflow from the last four sets of cones passes to dewatering tanks and is then thickened in Dorr machines. This section

of the plant will eventually conform to the following arrangement which handles 2800 tons per day.

Rod mill pulp goes to Dorr bowl classifiers. These are used instead of dewatering cones and provide the feed for the secondary grinders—Marcy ball mills. The ball mills are in closed circuit with Clark-Todd amalgamators and Dorr rake classifiers, from which 1200 tons goes to Dorr bowl classifiers. These effect separation of the sand and slime at about 200 mesh for cyaniding separately.

Stamping and milling cost 27.6 cts. per ton.

Cyanidation.—The next step is treatment of the sand and slime. The sand assays \$2.50 per ton and is all finer than 50 mesh, and 53 per cent passes 200 mesh. It is cyanided by aerating, leaching and washing and is finally sluiced out of the vats. The slime, 99 per cent of which is finer than 200 mesh, is treated in Merrill presses.

Precipitation.—The gold is precipitated from all solutions by zinc dust, the Merrill-Crowe process being used.

Extraction and Cost.—The total extraction is almost 94 per cent—62 per cent by amalgamation, 21 per cent by sand treatment and 10 per cent by slime treatment. The total treatment cost is 50 cts. per ton. The consumption of mercury is 0.128 oz.; cyanide, 0.27 lb.; zinc dust, 0.06 lb.; and lime, 3.12 lb. per ton ore treated.

Morro Velho Treatment.—The following information pertaining to Morro Velho practice was supplied by A. H. Millett, manager and superintendent, and is also from the bulletin "Reduction Works Practice at Morro Velho, Brazil," by Jack H. French and Harold Jones.

At the mine of the St. John del Rey Mining Company, Ltd., at Morro Velho in Brazil, the pyrrhotite has been for years the stumbling block, and it is still so little understood as to be capable, on occasion, of playing "tricks" without warning.

References to the action of pyrrhotite on cyanide solutions are rare in standard textbooks on cyanide metallurgy. If mentioned at all, it is often included with pyrite and marcasite under the heading of "iron sulphides," although its effects are even worse, owing to its ready liability to oxidation. . . .

The important minerals in Morro Velho ore are 20 per cent silica, 32 per cent iron, 15 per cent sulphur, 3½ per cent arsenic, 18 per cent

lime and $7\frac{1}{2}$ per cent magnesia. It consists of a finely crystalline admixture of quartz, calcite, dolomite, siderite, ankerite, pyrrhotite, pyrite and arsenopyrite with some chalcopyrite and traces of other metals. . . .

Pyrite and arsenopyrite are the only minerals present in any definite crystalline form, as visible to the naked eye, whereas the other minerals occur as a fine-grained mass, chalcopyrite having a tendency to segregate. . . . Pyrrhotite is the predominating sulphide, readily recognizable by its rapidly tarnishing to all shades of steel blue, pinkish red, and yellowish-brown. Chalmersite, a copper pyrrhotite, was first recognized here. . . .

The gold content of the ore averages 12 dwt. per long ton, and the silver content 4 dwt. The metals occur in a finely divided condition as a native alloy of 77 per cent gold and 23 per cent silver. Visible gold is rare. . . .

Microscopic examination of polished surfaces of the ore under reflected light reveals that gold is most generally associated with arsenopyrite and in contact with pyrrhotite . . . similar to the relation of gold to pyrite and pyrrhotite mentioned by Paige in *Bul. 765* (1924) of the United States Geological Survey.

Pyrrhotite, which has the general formula $\text{Fe}_m \text{Sm} + 1$, differs in composition from iron pyrite and many other sulphide materials, inasmuch as one sulphur atom appears to be loosely held in chemical composition and is easily capable of forming additive compounds, such as sodium thiocyanate, NaSCN , from cyanide. (On "weathering," the mineral readily yields elemental sulphur.) The $[\text{FeS}]$ remaining is particularly prone to oxidation, forming ferrous and ferric sulphate, which interact with cyanide to form complex cyanides. These reactions show that pyrrhotite is not only a powerful cyanide but also tends to rob the cyanide solution of much of the oxygen necessary for gold dissolution.

. . . The extreme susceptibility of pyrrhotite to rapid oxidation is well-known, this being enhanced by its frequent occurrence as minute crystals, which disintegrate to a substantial extent in the early stages of grinding, so that the latent acidity develops quickly and is difficult to control satisfactorily later, unless contra measures are taken.

Ores containing pyrrhotite are always difficult to treat satisfactorily by cyanide owing to the easily decomposable nature of that mineral. Pyrrhotite, if kept dry, is stable but in most atmospheres breaks down rapidly, and in contact with water the rate of decomposition is still more accelerated. The action is essentially one of part oxidation, and the products are not dissimilar from those which occur when pyrite or marcasite are weathered. . . . The main difference is that the rate of decomposition of pyrrhotite is markedly greater than that of the other common pyritic minerals . . . and larger quantities of ferrous com-

pounds are formed and to be dealt with than is usually the case with nonpyrrhotitic ores.

Ore treatment at Morro Velho is complicated. About one-half of the gold is liberated in grinding as finely divided metal but not amenable to amalgamation; hence an elaborate system of concentration is employed to recover the maximum amount before cyanidation. By concentration 58 per cent of the gold is saved, and by cyanidation 32 per cent. The ore average is 12 dwt. per long ton in gold and 4 dwt. in silver. The cost of treatment is $68\frac{1}{2}$ pence (\$1.37) per ton. Details of individual operations follow.

Crushing.—First-stage breakers reduce the mine ore to 5 in. Grizzly undersize and the crusher product pass through a trommel which delivers 2-in. ore to a picking table, where 4 per cent is rejected. Second-stage breakers reduce the large pieces to 2 in. The 140 stamps crush 5 tons a day each. With 30-mesh screens, the duty is 7 tons. A sieve test of the stamp-mill product gives 6.5 per cent on 60 mesh, 10.5 per cent on 100 mesh, 27.1 per cent on 200 mesh and 55.9 per cent through 200 mesh.

Concentrating.—The stamp-mill product is run over triple-deck canvas strakes set at a slope of $1\frac{3}{8}$ in. per ft.; by an intricate system of so-called "concentration" and "reconcentration" about 60 per cent of the gold is recovered at a cost of only 2 to $2\frac{1}{2}$ pence (4 to 5 cts.) per ton ore milled.

Classifying and Grinding.—Spitzkasten classify all the pulp from the canvas strakes. Cones thicken the coarse sand for grinding in tube mills. Of the tube-mill product, 65 per cent is finer than 200 mesh, and it is all passed over canvas to catch any gold that has been released in grinding. All pulp is finely separated into sand and slime in V-boxes and run to separate settling tanks.

Cyaniding.—Thick pulp from the sand settlers or charges and from the slime tanks is taken to fourteen 40-ton mechanical agitators provided with a low-pressure air supply. Cyanide is added here, bringing the solution strength to 0.27 per cent NaCN. Agitation proceeds for 12 hr. The sand and slime are next water washed on leaf filters, sands for 35 to 50 min. and slime for 40 to 95 min. If the sand and slime are mixed and filtered, the cycle is intermediate.

As the washed sand from the filters may contain some gold particles that are incompletely dissolved by cyanide, it is passed over another series of canvas strakes. The concentrate caught amounts to 9 tons a day, 8 dwt. per ton being assayed. This material contains 12 per cent arsenic, and it is either roasted or returned directly to the head of the cyanide plant.

Tailings from the reconcentration strakes that are actually concentrates from which practically all free gold has been removed are agitated for 12 hr. with cyanide in charges of 15 tons. This material assays 3 to 4 oz. per ton. The pulp is run into a large vat with a filter bottom connected to a vacuum receiver. Since the residue assays 0.2 oz. or more, it is roasted, and the calcine returned to the mill circuit.

Roasting.—Two wood-fired Edwards roasters, each with a capacity of from 8 to 10 tons per day, are required. About 200 tons refined arsenic is produced per year from the roaster fumes.

Gold Recovery.—Gold in solution is precipitated on "ribbon" made locally from cheap zinc containing 1 to $1\frac{1}{2}$ per cent lead. Although solutions are foul, precipitation is good, aided by drips of strong cyanide and lead acetate. All of the zinc-box material is treated with sulphuric acid. After being washed and dried, the bullion is melted in oil-fired graphite crucibles. The cyanide bullion and the rich concentrate bullion are chlorinated, yielding gold 996 fine.

SULPHOTELLURIDE ORES

Cripple Creek Ore. Golden Cycle.—At Colorado Springs the custom plant of the Golden Cycle Corporation has operated since 1907. The gold ores that are treated by roasting and cyanidation are the sulphotellurides received almost exclusively from Cripple Creek, according to L. S. Harner in *I.C. 6739, U. S. B. of M.*, 1933. Two types of ore come from the district, siliceous and basic, containing 76 to 87 per cent silica, $3\frac{1}{2}$ to 4 per cent iron, 1.8 to 2.3 per cent sulphur, 1.6 to 5.2 per cent lime and 0.5 to 1.4 per cent magnesia. The siliceous or oxidized ores are easily treated, but the denser and harder ores from depth give more difficulty. Cripple Creek ores of average grade (8 dwt. gold) are treated according to the flow-sheet in Fig. 124,

and low-grade ores as in Fig. 125. The latter applies also to the treatment of flotation tailings received from a section of the

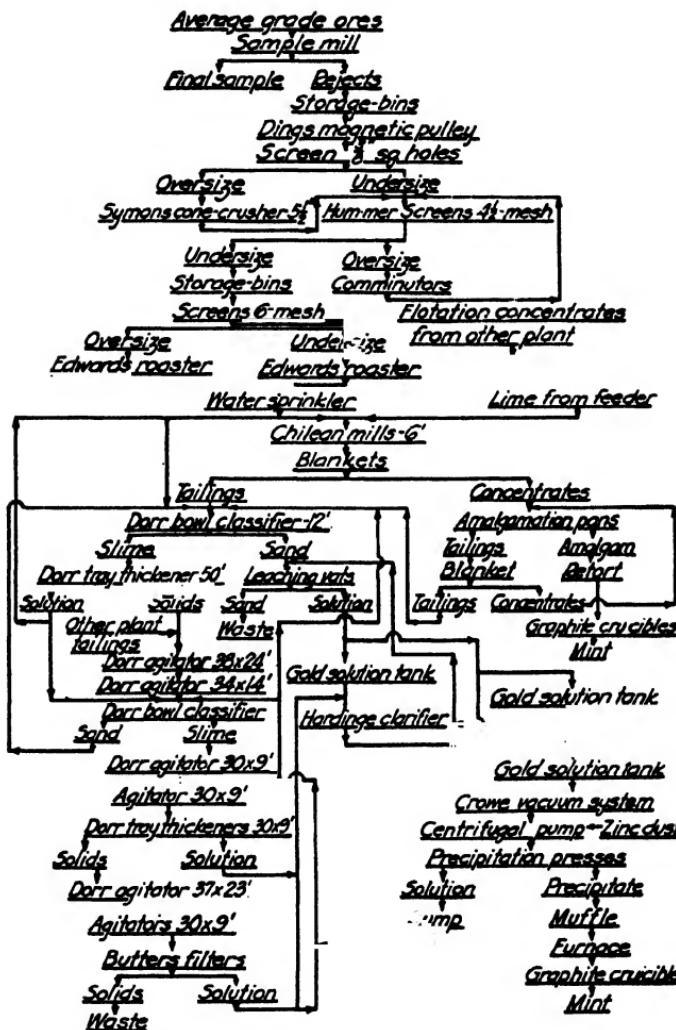


FIG. 124.—Flow-sheet for average-grade Cripple Creek ores.

plant that dresses copper, lead and zinc ores. All of these ores are carefully sampled and assayed.

Roasting in the Golden Cycle plant presented an interesting problem according to Harner in *E. and M.J.*, Jan. 28, 1928.

Basic or limey ores that contain 1.75 per cent sulphur were easily roasted and treated, but those with more than 2 per cent gave trouble in roasting, involving mechanical difficulties, higher

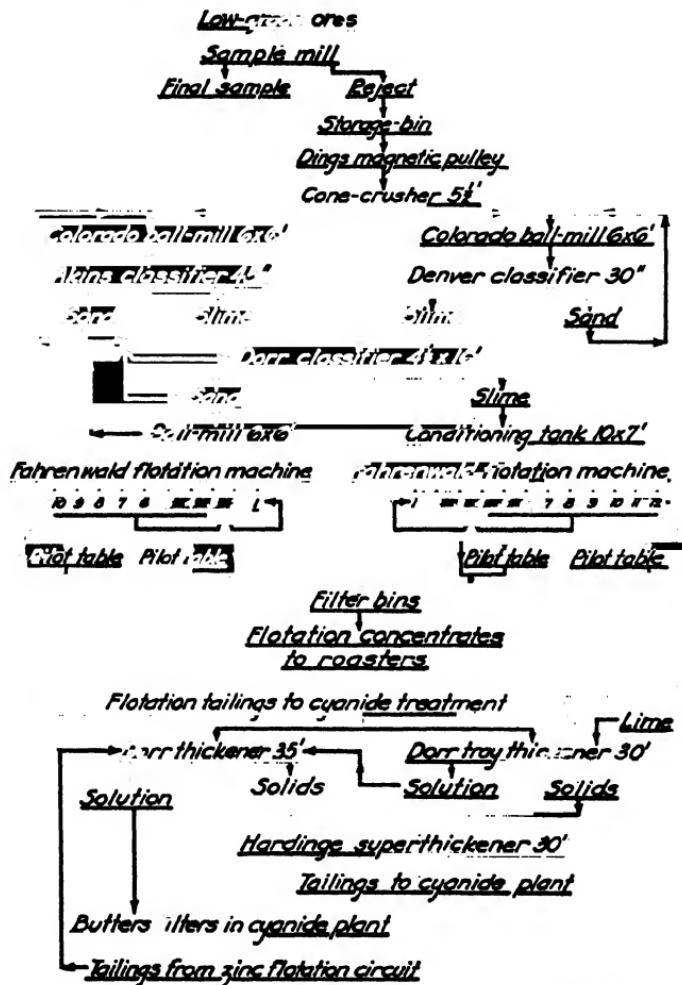


FIG. 125.—Flow-sheet for low-grade Cripple Creek ores.

consumption of chemicals and zinc and lower extraction of the gold. The reason for poor roasting of basic sulphur ores was not understood until a CO₂ recorder was installed at the rear end of one of the Edwards furnaces. With this class of ore the indicator rose until stopped by its limit. The roasting tempera-

ture was 850 to 900°C., and this produced CO₂ so copiously that the air supplied was almost entirely prevented from mixing with the freed sulphur, thereby precluding the oxidation of the sulphur. Large quantities of sulphides and sulphates remained, bringing their attendant ill effect in cyanidation. Temperatures of 450 to 500°C. were tried, and these proved high enough to drive off and oxidize the sulphur, without decomposing the carbonates. The temperature was set eventually at 600 to 650 °C. checked by indicating pyrometers and recorders. As a result, less coal was burned, furnace upkeep was lower and average residues were lower.

After cooling, the roasted ore is sprayed with water and delivered to Chilean mills which grind it in cyanide solution. The pulp flows over blankets, which are essential for good recovery. Then it is delivered to a Dorr bowl classifier, operated with liberal backwash water added to the raking compartment. This gives a sand practically free from slime, 85 per cent of it coarser than 60 mesh and assaying 0.24 dwt. oz. per ton.

The sand from the bowl classifiers is leached. The slime is thickened in a 50-ft. Dorr thickener, discharged by Dorco diaphragm pumps, delivered to a continuous Dorr agitator and finally filtered in Butters filters. Lime-plant solutions are clarified in a Hardinge sand clarifier. Gold is precipitated on zinc dust in the usual manner, after deaeration of solutions. Lead acetate is beneficial.

In treating more than 800 tons per day, amalgamation saves 25 per cent of the gold, and cyanidation 72 per cent at a cost of \$2.21 per ton. Chemical consumption is 0.96 lb. cyanide, 6.1 lb. lime, 0.17 lb. hydrochloric acid for filter leaves, 0.037 lb. lead acetate and 0.269 lb. zinc dust. (The plant has since been enlarged.)

Kalgoorlie Ore.—Ore of the “Golden Mile” near Kalgoorlie, Western Australia, occurs in quartz-dolerite greenstone, and there is little or no demarcation between country rock and ore, only careful sampling revealing the lode limits. The ore is fairly hard and carries an average of 5 per cent pyrite and a varying percentage of tellurides. Silica is high, and calcite is a constituent.

Kalgurli Ore Treatment Company.—Probably Kalgoorlie has been the outstanding exponent in the use of bromocyanide in

the treatment of sulphotelluride gold ores. Millions of tons of ore was treated in the plants of the Hannan's Star, Oroya-Brown Hill and Golden Horseshoe Companies, now liquidated. But all of these residues have been profitably treated since that time, by simple washing and filtering. At the present time, on the Boulder Perseverance mine is the plant of the Kalgoorlie Ore Treatment Company, Ltd., owned jointly by Boulder Perseverance, Ltd., and North Kalgoorlie (1912), Ltd. This recently enlarged plant has a daily capacity of 450 tons or more and employs bromocyanide. Near by, on the South Kalgoorlie, are an all-roasting and cyanidation mill and the all-flotation and concentrate-cyanidation mills of the Great Boulder Proprietary and Lake View and Star. A comparison of results of methods used at the Kalgoorlie Ore Treatment and Lake View and Star shows similar gold recovery—92 per cent—but costs of the latter are lower, although the former expects to reduce its charges considerably.

Ordinary cyanide solutions give a low extraction of gold from any raw ore in which the gold is locked in pyrite or exists as a telluride, but when bromocyanide is added, the gold and silver are dissolved rapidly. No oxygen is needed for the reaction



Bromocyanide alone does not dissolve the precious metals.

The practice at the Kalgoorlie Ore Treatment Plant is shown by the flow-sheet in Fig. 126. After the ore has been broken by a Hadfield jaw crusher, Gates gyratory crushers and a Symons cone crusher to $\frac{3}{16}$ in., Hardinge and Marey mills in closed circuit with Dorr classifiers in weak cyanide solution produce a final product of 90 per cent through 150 mesh. In the circuit are corduroy tables which catch 20 to 30 per cent of the gold. The final pulp after thickening in Dorr machines advances to mechanical agitators. In the agitators the cyanide solution is strengthened to 0.1 per cent, and agitation proceeds for 4 hr. Then the bromocyanide is introduced, with agitation for another 2 hr. Alkalinity is maintained at 0.0014 per cent lime by adding dilute sulphuric acid, but after agitating it is increased to 3 lb. per ton ore. (The reasons for low alkalinity when BrCN is being used are discussed under the section on testing.) Washing

with weak solution and water and drying of the cake are done on two Oliver and one Paxman filter. An interesting feature of water washing and repulping for pumping is that one lot of water costs 6 shillings (\$1.44), and the other 18 pence (36 cts.) per 1000 gallons, these being the charges made by the state government, respectively, for treatment water and for residue water. Gold is precipitated by zinc dust, and the sludge is fairly high in tellurium.

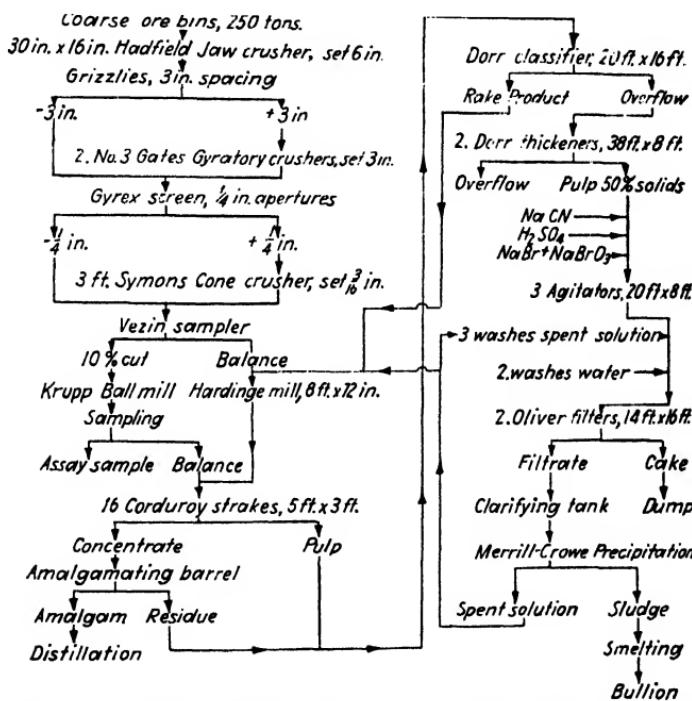


FIG. 126.—Flow-sheet of Kalgoorlie Ore Treatment plant, Western Australia.

Bromocyanide is made in the plant by adding sodium cyanide and sulphuric acid to sodium bromide and sodium bromate. This is done in a special tank, precautions being taken to avoid asphyxiation of the operators. The reaction causes a rise in temperature of the ingredients, so ice, also made in the plant, is added to keep the temperature down to 150°F. Consumption of bromo salts is 1 lb., and of cyanide $1\frac{1}{2}$ lb. per ton.

Lake View and Star.—In *M.M.*, October, 1933, T. B. Stevens describes in detail the scheme of ore treatment in the Lake View

and Star mill at Kalgoorlie. The present flow-sheet is the third that has been used on this complex sulphotelluride ore.

The ore is described as a sulphide containing (a) free gold which is soluble in cyanide and can be also saved by straking or amalgamation; (b) tellurides of gold which yield only to intensive cyanide treatment in the raw state but readily after roasting; (c) gold associated with pyrite, which will not give a satisfactory extraction without roasting.

The first flow-sheet employed gravity concentration, roasting and cyaniding of the concentrates with further cyanidation of the concentration tails. Gravity concentration was inefficient, and hence, when higher grade ore was treated, the over-all extraction was poor.

The second flow-sheet embodied dry grinding and roasting of the entire feed followed by cyanidation. Extraction was good, but costs were high because of the dry grinding and fuel required to roast an ore of relatively low sulphide content.

The present and third plant (Fig. 127) handles about 1500 tons per day, and a finished concentrate is made by flotation, the concentrate roasted and cyanided thereafter.

CRUSHING.—Primary crushing is in two No. $7\frac{1}{2}$ Gates gyratory crushers, producing a minus 5-in. product. The ore is given a secondary crushing in two 4-ft. Symons cone crushers, and oversize is finally crushed in a tertiary 4-ft. Symons cone. Mill feed is all minus $\frac{1}{2}$ in.

GRINDING AND GRAVITY CONCENTRATION.—Grinding is done in three stages, the first stage employing five 6- by 5-ft. open-circuit ball mills. The second stage uses three 5- by 11-ft. and two 5.5- by 22-ft. tube mills each in closed circuit with a 6-ft. by 25-ft. Dorr duplex classifier. The third stage is accomplished in a 5.5- by 11-ft. tube mill in closed circuit with two 6-ft. wide by 14-ft. diameter Dorr bowl classifiers. The final grind is to 95 per cent through 200 mesh.

Each secondary tube-mill discharge flows over four corduroy strakes 6 ft. long by 3 ft. wide. The tertiary mill has 10 strakes 10 ft. long by 3 ft. wide. An average of 24 per cent of the gold is concentrated on the corduroy.

FLOTATION.—The bowl overflow is thickened to 30 per cent solids in three 50- by 8-ft. Dorr thickeners. Roughing is done by three No. 24 Fahrenwald Sub-A flotation machines. The first

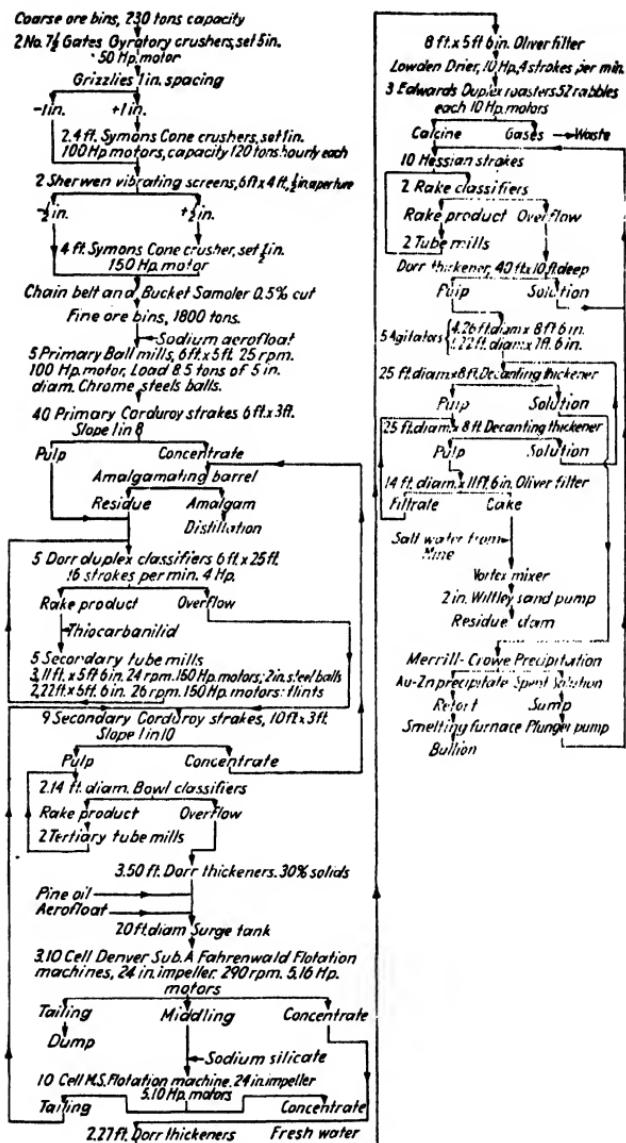


FIG. 127.—Flow-sheet of Lake View and Star plant, Western Australia.

cell froth is taken as finished concentrate, and the remaining froth goes to a 24-in. Minerals-Separation machine for cleaning. The cleaner tails return to the bowl classifiers.

As the quantity of pyrite seldom exceeds 5 per cent, the rougher froth is small in bulk and fragile. The standard Fahrenwald machine was altered somewhat to make it easier to remove the froth. A conical shell, 4 ft. square at the base and with a 2-ft. diameter opening, fits down in the cell and thus reduces the overflow area. Aprons attached to the hood drain the froth to the side of the machine. This crowding of the machines increases capacity and makes for easier control.

Flotation reagents used are as follows:

Reagent	Where used	Pounds per ton
Sodium Aerofloat.....	Ball mills	0.04
Thiocarbonalide.....	Tube mills	0.05
Aerofloat 25.....	Flotation surge tank	0.005
Pine oil No. 5.....	Thickener underflow	0.04
Sodium silicate.....	Cleaner feed	0.15

It was noted that xanthates produce a froth too dry to handle easily on this ore, thiocarbonalide producing a much tougher froth. Sodium silicate acts as a gangue-depressing agent. The concentration ratio is in the neighborhood of 15 to 16 to 1.

FILTRATION AND DRYING.—The finished concentrate is thickened in a 20- by 8-ft. thickener and filtered on two 5-ft. 8-in. by 8-ft. Oliver filters. Moisture is reduced from 12 to 7 per cent in two Lowden dryers with an area of 480 sq. ft.

ROASTING AND CYANIDATION.—The moist concentrate containing 35 per cent sulphur is roasted in three duplex Edwards furnaces. The burning takes place freely without auxiliary fuel. Successful roasting depends upon slow oxidation with ample air.

The cooled calcine is ground in solution in a 4- by 4-ft. ball mill and then given 18 hr. batch agitation at 2 to 1 dilution in five 26- by 8-ft. stir agitators. Batch agitation is preferred to continuous so that poor roasts may be isolated and given preferential treatment. The agitator discharge is decanted twice in two 25- by 8-ft. thickeners and filtered on an 11-ft. 6-in. by

14-ft. Oliver filter. Water and barren solution are used as wash. Precipitation is by the Merrill-Crowe process.

Over-all extraction of the mill is 91.7 per cent, 24 per cent by gravity concentration and 67.7 per cent by flotation and cyanidation. Cyanide extraction of the concentrates is 98.3 per cent. All of this is on mill heads assaying 35 shillings (\$8.50) per ton.

Present treatment costs are given as 6 shillings 4 pence (\$1.52), but these will be reduced when part of the old dry-crushing plant still operated is shut down. Water costs are high, amounting to 1 shilling 4 pence (32.3 cts.) per ton feed.

South Kalgoorli.—The South Kalgoorli plant is the only remaining example in Kalgoorlie of the all-dry crushing, roasting and filter-pressing process for sulphotelluride gold ore. At one time, plants of this type were treating more than 2000 tons a day. All have been superseded by flotation, except one bromocyanide plant. If the South Kalgoorli had a larger ore reserve, the process probably would be changed, but the present system extracts 95 per cent at reasonable cost.

The flow-sheet of the South Kalgoorli plant shown in Fig. 128 is taken from the article by G. B. O'Malley in *C.E. and M.R.*, January, 1934. The plant treats 330 tons a day of ore which averages 10 dwt. gold per ton and 6 per cent pyrite and tellurides. The ore as it comes from the mine is dry enough so that it can be crushed through about 28-mesh screens in Krupp ball mills. Each of the four mills is loaded with $2\frac{1}{2}$ tons of 5-in. balls and crushes 90 tons ore per day. Hot air from the roasting-furnace flue is conducted to the mills to aid grinding. Roasting is done in 10 Merton three-hearth furnaces and two Edwards duplex furnaces. The former roast 20 tons each, and the latter 75 tons each per day. Local wood is used for fuel in the roasting furnaces. Roasting costs 4 shillings (96 cts.) per ton. The roasted ore contains 1.65 per cent total sulphur. It is mixed with weak cyanide solution and fed into grinding and amalgamating pans in which 25 per cent of the gold is saved. The overflow from the pans goes to classifying cones, the sand from which is ground further in a tube mill. The mill discharge flows over 48 sq. ft. of corduroy set at a slope of 1 in 6. All of the slime is agitated and aerated for 8 hr. with 0.05 per cent cyanide solution. Filtration is done in Dehne filter presses. The residue is pumped to a distant dump. Prior to precipitation, the solution strength is

raised to 0.06 per cent cyanide, and 0.015 lb. lead nitrate per ton solution is added. Lead is also added to the hot pulp from the roasting furnaces. The Merrill-Crowe process of clarifying, deaerating and precipitating the gold is satisfactory, and the

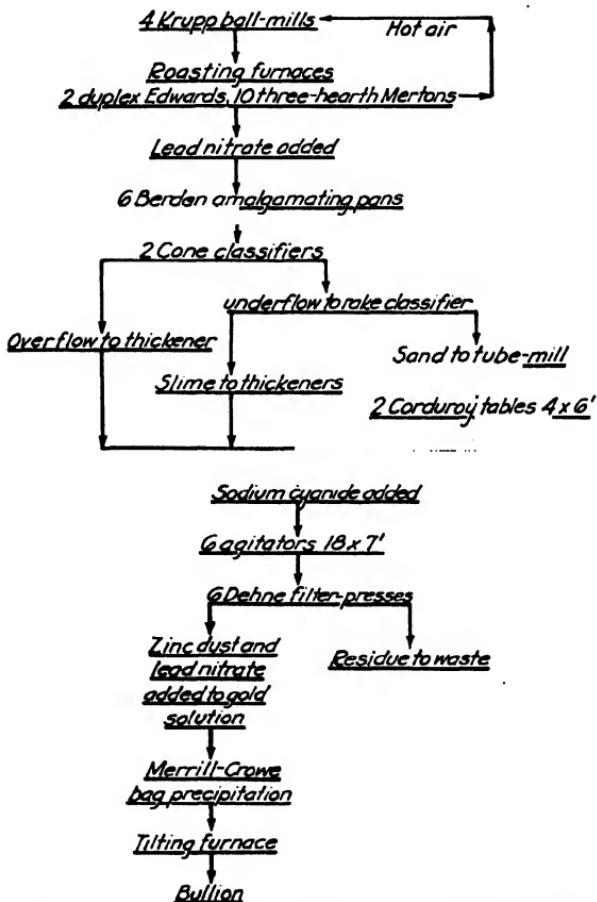


FIG. 128.—Flow-sheet of South Kalgoorli plant, Western Australia.

consumption of zinc is low. The bullion contains 6 to 12 per cent silver and 2 per cent base metals. Cyanide consumption is only 1 lb. per ton.

CHEMICALS USED AT KALGOORLIE.—Consumption of chemicals in cyaniding Kalgoorlie ore is approximately as follows: NaCN,

1 to $1\frac{1}{2}$ lb.; BrCN, 1 lb.; CaO, 4 lb.; Pb (NO₃)₂, 0.15 lb.; and zinc, 0.05 to 0.3 lb. per ton ore treated.

Solubility of Tellurides.—In *Bul. C.I.M. and M.*, June 1933, under the heading "Gold Tellurides Are Soluble in Cyanide," W. E. Johnston wrote:

The gold-bearing tellurides behave very much as does gold itself. Finely divided particles are dissolved fairly rapidly, coarser particles more slowly. All tellurides are soft and brittle and also have high specific gravity, with the result that they are extremely finely divided in ordinary mill practice. The ordinary high-lime solution of Kirkland Lake practice is suitable for dissolving of the gold tellurides. Oxygen is necessary but apparently not more so than in dissolving gold.

In *E. and M.J.*, August, 1933, Johnston returns to the problem with the results of experiments. Before detailing these he says that a search of the literature failed to show that anyone had actually conducted cyanidation tests on either specimen tellurides or those concentrated from an ore--meaning Kalgoorlie, Cripple Creek and Kirkland Lake. Experiments with cyanide on these ores gave the following conclusions, with which A. L. Blomfield, H. V. Wallace and John Dixon of Kirkland Lake substantially agree:

1. Gold-bearing tellurides do yield their gold to cyanide, if they are in a finely divided state and excess lime is used.
2. Sodium peroxide greatly reduces the time of treatment required for maximum extraction. It is not beneficial when used in quantities equivalent to commercial use.
3. The tellurides are very brittle and, owing to their high gravity, will be retained in the mill circuit for a long time. They will thus be in a finely disseminated state, approximately minus 1600 mesh or the size required to yield a maximum extraction of their gold.
4. Up to the present, gold-bearing tellurides have not been found in large quantities in mill tailings or in concentrates recoverable from them.

Proof enough that cyanide solutions dissolve tellurides is shown in analyses of precipitate, bullion and matte from melting at Kirkland Lake. Wright-Hargreaves precipitate carries 6 per cent or more of tellurium, also 16 per cent copper and 9 per cent zinc. A process is being perfected to extract the tellurium from the precipitate, some already having been recovered and melted. Lake Shore bullion contains about $1\frac{1}{2}$ per cent tellurium, also $4\frac{1}{2}$ per cent copper, 2 per cent zinc and 2 per cent

lead. During a pour, fumes of tellurium could be seen arising from the mold. Matte at the Toburn contains tellurium but if simple is melted with cyanide to recover the gold content.

GOLD RECOVERY FROM BASE-METAL ORES

In general, the recovery of gold and silver from ores containing copper, lead and zinc is not so high as desired; recently, equipment has been added in several plants dressing such ores to increase the recovery of the precious metals. Descriptions of the processes follow:

GOLD-COPPER AND COPPER-GOLD ORES

Some mines produce a gold-copper ore in which the copper is of subsidiary importance; others produce a copper-gold ore in which the gold is of minor importance. However, in some of the latter types enough free gold is present to warrant special equipment for its recovery. Mount Morgan and Tul Mi Chung are examples of the first type of ore, and Britannia represents the second type.

In *T.P. 410, A.I.M.E.*, 1931, E. S. Leaver and J. A. Woolf devote 32 pages to the flotation of minor gold in large concentrators. In many copper-milling operations, with the gold content under \$1 per ton, only half of the gold is saved in the flotation concentrates. Lime is usually used as a depressant for pyrite in the flotation circuit, and high lime or a solution high in calcium ions is also a depressant for gold. Their experiments on ore from the Walker, Utah, Copper, Nevada Consolidated, Old Dominion, Magma and United Verde mines showed that in the flotation of the lime, avoiding excess will increase the gold recovery. By control of the flotation circuit, low alkalinity thus being assumed, ores with a low ratio of pyrite to copper sulphide and containing less than \$1 free gold per ton gave a concentrate containing over 90 per cent of the copper and 80 per cent of the gold. When lime is used the optimum range of alkalinity for the flotation of gold is 7.5 to 10 pH.

Gold-copper. Mount Morgan Practice.—At Mount Morgan, Australia, the gold-copper ore is dressed as follows, according to A. A. Boyd, general manager: The plant was handling 500 tons a day but has been increased to 700 tons. The ore is a highly pyritic quartzite, partly oxidized, and assays 6 dwt. gold per

ton and 5 dwt. silver per ton, also 0.6 per cent copper in chalcopyrite. Tellurides are also present in small amounts. The mine has reserves estimated at several million tons of lower gold content and higher copper content. The plant layout includes a 36- by 24-in. jaw crusher set at $4\frac{1}{2}$ in. followed by gyratory crushers set at $1\frac{1}{2}$ in. and rolls in closed circuit with Leahy screens which give a $\frac{1}{8}$ -in. product. Five ball mills in closed circuit with five Dorr classifiers make 55 to 60 per cent minus 200-mesh product, and two Dorr bowl classifiers discharge 80 to 85 per cent through 200 mesh. The pulp is conditioned, and the minerals are floated in subaeration and Fagergren cells. The concentrate, which is $2\frac{1}{2}$ per cent of the mill feed, is thickened, dried to 20 per cent moisture on a disc filter, heat-dried, bagged and sold to a smelter.

Tul Mi Chung Practice.—Flotation is the only treatment to which the gold-copper ore of Tul Mi Chung mine, Korea, would yield satisfactorily. Part of the gold is free, some is combined with chalcopyrite and an iron-arsenic mineral, and some with pyrite, according to R. J. Lemmon in *Bul. 229, I.M. and M.*, 1923. The loss of gold was traced to the finely divided pyrite, much of which escaped with the tailings. Modified Minerals-Separation flotation machines were adopted, and recovery is satisfactory. The concentrate carried 5 oz. gold and 6 oz. silver per ton and 25 per cent copper. A large dump of 2.5 dwt. gold and 0.14 per cent copper tailings, not amenable to cyanidation, was treated by flotation.

Copper-gold. *Britannia Practice.*—Minerals of the Britannia Mining and Smelting Company, British Columbia, are mainly chalcopyrite and pyrite in a quartz-porphyry gangue. One section of the mine yields ore assaying 0.6 per cent copper and 0.86 dwt. gold per ton, according to H. A. Pearse in *M. and M.*, September, 1934. Some of the gold is free. To collect the coarser particles of gold which are difficult to float, the pulp overflowing each classifier is passed first over a 3- by 20-ft. wool blanket. The blankets are taken up and washed once or twice a shift. An iron-zinc sulphide tailing from the secondary copper flotation is passed over blankets to recover any remaining coarse particles of gold. As a further provision against loss of gold, all tailings from the flotation plant are run over 4000 sq. ft. of blankets laid in shallow launders 30 in. wide. The blankets rest

on discarded screens from the crushing plant, $\frac{1}{2}$ in. above the bottom of the launders, set at a fall of $1\frac{1}{2}$ in. per ft. Each blanket is washed in place by a high-pressure hose. Total recovery of gold, both free and in copper minerals, is 60 per cent. The launders catch several per cent at low cost.

COPPER-ZINC ORE

Flin Flon Practice.—During 1932 the Flin Flon property of the Hudson Bay Mining and Smelting Company, Manitoba, yielded 82,565 oz. gold and 933,983 oz. silver, as well as copper and zinc, from 1,439,651 tons ore. The flotation plant makes a copper concentrate, a zinc concentrate and a tailing for cyanidation which involves the following process, according to G. E. Cole in the *Bul. C.I.M. and M.*, February, 1932:

Following the recovery of copper and zinc sulphides by flotation, a considerable recovery of gold and silver is made in the cyanide plant, where the heavy sulphide flotation tailings, after being thickened in a 90-ft. Dorr thickener, are pumped to Devereux agitators, 18 ft. diameter and 18 ft. high where 4 lb. of cyanide per ton of solution is added. The pulp, in flowing through, is constantly agitated and is then passed to Dorreco filters, 14 ft. diameter and 18 ft. long. The final cake from the filter treatment joins the tailings from the disseminated-ore flotation section, and the thickener overflow, and the whole is pumped to a tailings-disposal area.

To the pregnant solution, after clarification, zinc dust is added, and the gold-silver precipitate goes to the Merrill presses. This precipitate is next treated with acid to dissolve the excess zinc dust and, when dried, is melted into bullion, later to be refined. The barren solution from the cyanide treatment is delivered for regeneration of the cyanide, being mixed with acid or water containing sulphur dioxide and copper sulphate. The cyanide is removed by air and absorbed in lime solution, the units being in absolutely tight equipment. The acidified solution from which cyanide has been removed is next pumped to Shriver presses, from which the copper sulphocyanide is removed as a precipitate which, after roasting to a copper sulphide cinder, is transported to the copper smelter bedding bins.

MIXED COPPER, LEAD AND ZINC ORES

Golden Cycle Practice.—The Golden Cycle mill at Colorado Springs, Colo., receives, in addition to the regular Cripple Creek ore, complex copper, lead and zinc ores. The following informa-

tion is from an article by Max W. Bowen in *E. and M.J.*, August, 1935.

A separate flotation concentrating plant of about 450 to 475 tons daily capacity (Fig. 73) has been built to handle these ores. The general treatment is to remove free gold in unit cells and make a lead-copper concentrate and an iron-zinc concentrate. The lead-copper concentrate is shipped to the Leadville refinery, and the iron-zinc concentrate is roasted and fed with roasted ore in the cyanide plant. Flotation tails are returned to the slime section of the cyanide plant.

Crushing.—Primary breaking is done in two 13- by 24-in. Farrel jaw crushers and two sets of 40- by 14-in. rolls. A 1 per cent sample is cut from 25 per cent of the roll product. The feed goes to a stationary screen with $\frac{5}{8}$ -in. square holes. The oversize is crushed in a 5.5-ft. Symons short-head cone crusher, and the undersize combined with the crusher product goes to four 4-mesh Hum-mer screens in closed circuit with another 5.5-ft. Symons short-head cone crusher and two 6- by 6-ft. Schmidt comminuters. Final mill feed runs 2 per cent plus $\frac{5}{32}$ in. and 61 per cent minus 10 mesh.

Grinding and Concentration.—Grinding is done in three 6- by 6-ft. Colorado Iron Works ball mills in closed circuit with a 4- by 4-ft. trommel with 0.104-in. openings. Free-gold is removed in two No. 500 Denver Equipment unit cells. The concentrate from these cells is amalgamated in a Wheeler pan, and the pan tails returned to the grinding circuit. The flotation tails go to a 6-ft. by 23-ft. 4-in. Dorr duplex multizone classifier, the sand being returned to the ball mills. The overflow is conditioned in two 10- by 8-ft. Denver Equipment conditioners and then passes to two No. 18 Denver Equipment flotation units where a lead-copper concentrate is produced. This concentrate is washed with cyanide solution in a 16- by 16-ft. thickener, filtered on a six-disc, 6-ft. American filter and shipped to the smelter.

An iron-zinc concentrate is removed in the third flotation unit of No. 24 Denver Equipment cells and also from six Wilfley tables. The combined concentrates are dewatered in a Stearns Rogers drag. The overflow is thickened in a 17- by 6-ft. thickener and then filtered.

The iron-zinc concentrate is added to the regular Cripple Creek ore ahead of the stationary screen in the crushing plant and

receives the usual roasting and cyanide treatment. The concentrate tails are returned to the slime section of the cyanide plant.

The cyanide plant (Cripple Creek), in which about 85 per cent of the Golden Cycle ores are cyanided, makes a sand-slime separation and treats each separately. The sand-leaching operation is described in Chap. VII, and the roasting operation in Chap. IX.

Fresnillo, Mex.—The mill of the Fresnillo Company at Fresnillo, Zacatecas, Mex., treats oxide ores from its mine, dump ores and underground ores from the Plateros district, company leases and custom ores from the Zacatecas district, custom ores from other districts and tailings from old dumps at Fresnillo. The normal capacity of the plant is approximately 2600 dry short tons per 24 hr. The final product of the mill is a precipitate containing gold and silver, which is smelted and shipped as doré bullion to a refinery, according to W. E. Crawford in *Trans. 112, A.I.M.E.*, 1934.

The valuable metals are silver and gold, the ratio by weight of the former to the latter being 1000 to 1.5. Argentite is the predominant silver mineral and is associated with pyrite and manganese mineral. The sulphides are primary throughout. A slightly secondary action is represented by small quantities of native silver and silver halides, principally cerargyrite, which are found in the upper part of the mine.

Treatment may be tabulated as follows:

1. Ores weighed.
2. McCully gyratory crusher set at $3\frac{1}{2}$ in.
3. Grizzlies with $1\frac{1}{2}$ -in. openings.
4. Two 48-in. Symons horizontal-disc crushers set at $1\frac{3}{4}$ in.
5. Leahy vibrating screens with $\frac{5}{8}$ -in. openings.
6. Two 48-in. Symons vertical-disc crushers set at $\frac{3}{8}$ in.
7. Bin of 6400 tons capacity.
8. Two units each of two $6\frac{1}{2}$ - by $2\frac{1}{4}$ -ft. Marcy rod mills and one 6-by 14-ft. Traylor ball mill and one unit of two Hardinge ball mills and one Traylor ball mill, each unit in closed circuit with an 8- by 20-ft. Dorr duplex bowl classifier.
9. One Traylor mill in closed circuit with bowl classifier grinding oversize from dump tailings being treated.
10. Two 6- by 26-ft. Bowl classifiers making sand and slime.
11. Sand leached with cyanide solution 10 days or more.

12. Slime thickened in six 60- by 10-ft. Dorr machines, agitated in 0.125 per cent cyanide solution for 5.3 hr. in two circuits of six and seven Dorr agitators 40- by 25-ft. using air and filtered in Butters tanks.

13. Silver and gold precipitated on zinc dust.

In the crushing plant, tramp iron is removed by a magnet and by a magnetic pulley. Dust is collected and is sludged prior to going to treatment. Cyanide consumption is 3.34 lb., and lime consumption 16.2 lb. per ton ore treated.

SILVER ORES

Pachuca, Mexico.—Silver in the Pachuca district, state of Hidalgo, Mexico, occurs chiefly as argentite. A part composite analysis of ore treated at the present time is given in the accompanying table.

ANALYSIS OF PACHUCA ORE

Constituent	Per cent	Constituent	Per cent
Silica.....	71.6	Copper.....	0.05
Alumina.....	8.7	Manganese.....	1.6
Iron.....	2.9	Sulphur.....	0.7
Lead.....	0.05	Lime and magnesia.....	3.8
Zinc.....	0.5	Sodium and potassium oxide.	1.9

Flotation has been given thorough trials but has not succeeded in equaling the economic results of cyanidation, according to R. R. Bryan and M. H. Kuryla in *Trans. 112, A.I.M.E.*, 1934.

Three well-known plants are operating in the district—Loreto, San Rafael and Santa Gertrudis. The Loreto plant of Compañía de Real del Monte y Pachuca has a daily capacity of 3800 tons and is the largest silver-cyaniding works in existence. No concentration is done. Some details of 1934 operations of the Loreto follow.

Crushing.—Mine ore of a maximum size of 12 in. is reduced to $\frac{7}{8}$ in. by one gyratory and two cone crushers, between which are grizzlies and vibrating screens.

Grinding.—Two-stage grinding in cyanide solution is practice. For primary grinding, 8- by 6-ft. Marcy grate mills and a 6- by

12-ft. trunnion Traylor ball mill are used, in closed circuit with 6- by 22-ft. Dorr classifiers, which are the only type operated in Pachuca. Nearly 80 per cent of the feed to these mills is coarser than 3 mesh and up to 1 in. The classifier overflow is 64 per cent solids. For secondary grinding, 6- by 10-ft. Traylor mills of the trunnion type and 5- by 10-ft. trunnion mills of local make are used in closed circuit with an 8- by 22 and a 6- by 22-ft. classifier, respectively. The classifier overflow contains 20 per cent solids; a sieve test of the final product shows on 48 mesh 2.20 per cent; on 65, 7.90; on 100, 9.17; on 150, 13.13; on 200, 8.09; and through 200, 59.51 per cent.

Thickening.—Nine Dorr thickeners, 48 $\frac{3}{4}$ by 15 $\frac{1}{4}$ ft., yield a pulp of 48 per cent solids.

Agitating.—Twenty Pachuca tanks, 15 by 60 ft. and 30 "Flat" tanks, 20 and 24 by 30 ft., do the agitating. The latter is a tank equipped with a Dorr-thickener mechanism and air jets. Air at 35 lb. pressure is used in the Pachucas and at 18 lb. in the Flat tanks. Agitation proceeds for 73 and 70 hr., respectively, and the cost is 14.7 centavos (4.1 cts. or 2 pence) per ton ore.

Aero-brand cyanide is dissolved in barren solution to make a strong solution, and this is added to the agitators to bring the strength to 0.25 per cent NaCN. Litharge is added in the dissolving tank to eliminate soluble sulphides. Cyanide consumption, excluding regeneration, amounts to 1.62 kg. per ton ore. Lime consumption is 6 kilograms.

Filtration.—Butters tanks, each with 187 leaves, 67 by 117 in., do the filtering. Each tank averages 11 cycles of 128 min. each day, and each cycle is divided into 20 min. for caking, 44 min. for barren wash, 5 min. for water wash to mill and 30 min. for water wash to regeneration, the remaining 29 min. being required for filling transfers, discharging, etc. A vacuum of 18 in. is maintained. Average cake is $\frac{7}{8}$ in. thick.

Clarification and Precipitation.—Solution from the filters is clarified in 11 Sweetland presses which can handle 2 $\frac{1}{4}$ tons per day per sq. ft. surface. They are discharged twice and cleaned once each day, and leaves are acid treated every 10 days.

The Merrill-Crowe system of zinc-dust precipitation is used. Centrifugal pumps force the solution through the presses. Zinc consumption is 630 grams per kilogram of silver precipitated.

The dried precipitate assays 83 per cent silver and 0.46 per cent gold, also 0.25 per cent selenium and some other metals.

Melting and Refining.—Precipitate is melted to bullion in the usual manner, granulated borax and bottle glass being used, in an oil-fired reverberatory furnace of 15 tons capacity. The temperature is raised to 1050°C., and slag is skimmed off. Air is then blown in, and the slag is skimmed for 60 hr. Then the metal is tapped into a continuous anode-casting machine. Anodes weigh 10 kilograms, and a furnace charge makes 2000 of them in 5 hr. of casting. The bullion is increased in fineness from 950 to 993; copper is the principal impurity remaining.

The anodes are next parted in 200 Thum-type electrolytic cells, and the resultant silver is 999 + fine. The gold mud is reduced to anodes, which are treated in Wohlwill cells, giving gold 999.8 fine.

Extraction.—The current extraction from ores assaying 400 grams (250 dwt.) silver and 2 grams (1.49 dwt.) gold is 90 per cent silver and 92 per cent gold. On "clean" ores this reaches 93 per cent silver and 95 per cent gold.

Treatment Costs.—Treatment costs at Loreto are not available, but the following costs for a 400-ton cyanide plant in the Pachuca district is of interest and is taken from the paper by Bryan and Kuryla (*Trans. 112, A.I.M.E.*, 1934) previously cited.

Item	Pesos per ton	Distribution	Pesos per ton
Crushing and grinding.....	1.27	Labor.....	1.38
Cyanidation.....	1.75	Supplies.....	2.04
Precipitation and melting.....	0.30	Power.....	0.72
Water supply.....	0.14	Miscellaneous.....	0.24
General expense.....	0.92		
Total.....	4.38	Total.....	4.38

Prevailing rate of exchange 1934, 3.60 pesos per U. S. dollar.

Rosario, Honduras, Treatment.—In Honduras, Central America, the New York and Honduras Rosario Mining Company is treating 450 tons 20-oz. silver ore daily. The ore also carries 1.4 dwt. gold, 0.5 per cent lead, 0.63 per cent manganese and 2

per cent zinc. Two gyratory crushers make a 2-in. product which is fed to twenty 1800-lb. stamps crushing through 1-in. screens. Cyanide is added at the stamps. Grinding is done in a 6- by 5-ft. Allis-Chalmers ball mill and in two stages of two 5- by 22-ft. Allis-Chalmers tube mills in open circuit with Dorr classifiers. Of the final pulp, 20 per cent remains on 150 mesh.

Four 35- by 10-ft. Dorr thickeners produce pulp of 60 per cent moisture. This is agitated for 80 hr. in fourteen 15- by 45-ft. Pachuca tanks and in three 35- by 10-ft. mechanical storage tanks. The air pressure in the Pachucas is 35 lb. per sq. in.; strength of cyanide 0.3 lb. per ton solution and of lime 0.4 lb. per ton solution. Consumption of cyanide is 3.8 lb., and of lime 18 lb. per ton ore treated.

Filtration is done in two Merrill pressure filters of 4- by 6-ft. size with one hundred 3-in. frames. A total of 500 tons barren-solution wash and 700 tons water wash is used per day. Dissolved metals in the residue are 9 cts. silver and 2 cts. gold per ton; and undissolved metals, 77 cts. silver and 13 cts. gold. Regeneration of the cyanide is practiced in this plant.

Precipitation is effected by zinc dust, the consumption of which is 0.7 lb. per ton ore. The pregnant solution averages \$4.29 per ton. The precious metal from 1200 tons solution is precipitated in 24 hr. The effluent carries only a trace of metals. Thirty-seven per cent silver and 76 per cent gold is dissolved before agitation, 52 per cent silver and 18 per cent gold during agitation and 0.3 per cent silver and 0.56 per cent gold in filter.

Shafter, Tex. (Presidio).—Since 1883, the Presidio mine of the American Metal Company, Shafter, Tex., has produced 1½ million tons ore which has averaged around 18 oz. silver per ton. It has been a regular and interesting producer. A 90 per cent recovery is being made in the 200-ton gravity concentration and c.e.d. cyanide plant which is well described by V. D. Howbert and F. E. Gray in *T.P. 368, A.I.M.E.*, 1930.

The ore is oxidized and siliceous, the principal constituent of the gangue being quartz. The greater part contains approximately 20 oz. silver per ton, but mixed with this are many particles of ore of much higher grade, frequently assaying 100 oz. or more. In the 20-oz. material is an almost constant amount of silver which is difficult to extract by cyanidation. The high-grade material contains a much greater proportion of silver than

is easily amenable to cyanidation treatment. Thus, as the grade of ore becomes higher, the percentage of recovery of silver increases rapidly, under equal milling conditions. The silver minerals contained in the ore are principally argentite and cerargyrite, the former predominating. The lead minerals, all of which are argentiferous, are chiefly cerussite and galena, with occasionally a little anglesite. The gold is free; most of the ore contains merely a trace, but occasional concentrations are as high as 0.5 oz.

The crushing plant, which includes vibrating screens, makes a product of which 23 per cent is minus 8 and plus 20 mesh, 28 per cent is minus 20 and plus 40 mesh, 11 per cent is minus 100 and plus 200 mesh and 17 per cent is through 200 mesh. Ten per cent of the ore is sorted as water. To rid the screen oversize and the tube-mill discharge of as much lead as possible, these products are passed over Wilfley tables. The concentrates are smelted. The two tube mills are operated in closed circuit with Dorr duplex classifiers which make a final product of 75 per cent through 200 mesh. The pulp is cyanided by agitation and C.C.D. and filtered on Oliver machines. The loss of dissolved silver is 1.4 dwt. per ton dry tailing. Precipitation is by zinc dust, following usual procedure.

Lime is fed ahead of the Hum-mer screen at the rate of 10 lb. for each ton ore. Accurate control is necessary. A 10 per cent aerated solution of Aero-brand cyanide is made up and added to the first and fourth Pachuca tanks. A free-cyanide strength of 2.7 to 3 lb. per ton solution is maintained in the tanks. Consumption of cyanide is 1.7 lb. and of zinc dust 0.36 lb. per ton ore. The cost of milling and treating averages \$1.90 per ton.

Tabling operations at Presidio are described in detail in the chapter "Concentration and Treatment of Concentrates."

Sunshine, Idaho.—The Yankee Boy mine of the Sunshine Mining Company, 6 miles from Kellog, Coeur d'Alene district, Idaho, is the largest producing silver mine in the United States. In 1934 it produced about 3,450,000 oz. silver and 850,000 lb. copper. Silver is the principal product and occurs with tetrahedrite, $Cu_8Sb_2S_7$. A new mill of 500 tons capacity was completed in February, 1935. It is expected that a recovery of better than 96 per cent will be made. The ore averages more than 40 oz. per ton from which a concentrate exceeding 900 oz.

per ton is made. Because of the antimony and arsenic content of the product, smelting charges are high. Milling costs for 1933 were 79 cts. (40 pence) per ton, but higher labor and supply costs are expected to increase this to 90 cts. (45 pence).

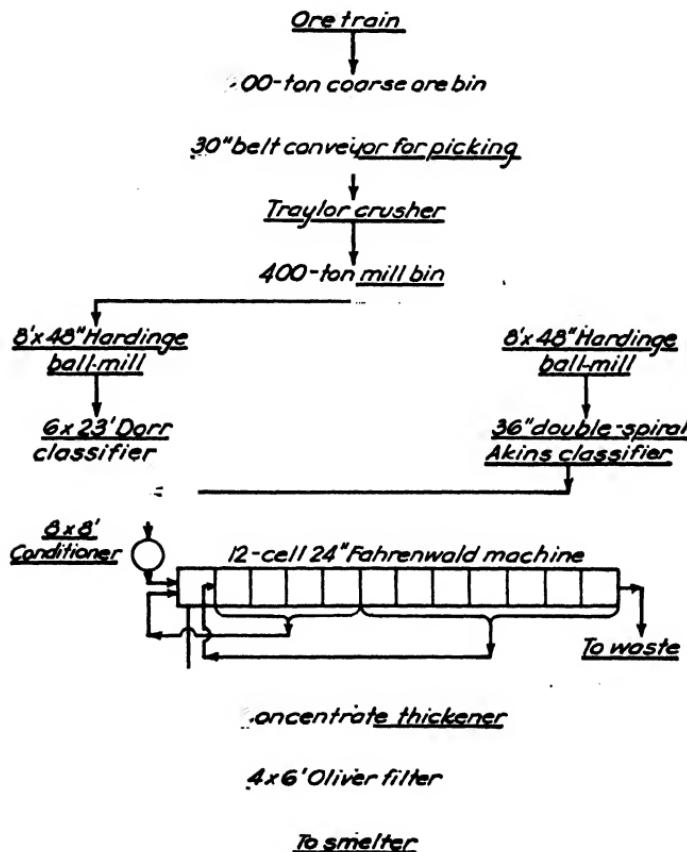


FIG. 129.—Flow-sheet of Sunshine silver-ore treatment plant, Idaho.

The simple flow-sheet of the new plant shown in Fig. 129 is self-explanatory. Sieve analyses of the ball-mill and classifier products are shown in the accompanying table.

The flotation reagents used in the old mill were 0.17 lb. per ton of a mixture of 40 per cent coal tar, 40 per cent Barrett No.4 oil and 20 per cent pine oil No. 5, with 0.05 lb. potassium ethyl xanthate.

SIEVE ANALYSIS OF BALL-MILL AND CLASSIFIER PRODUCTS, SUNSHINE MILL

Ball-mill discharge			Classifier		
Mesh	Per cent	Sand		Overflow	
		Mesh	Per cent	Mesh	Per cent
+14	2.3	+14	2.8		
20	4.6	20	3.7		
28	8.5	28	8.5		
35	8.5	35	10.7	+35	0.1
60	29.0	60	26.8	60	3.0
80	12.2	80	19.7	80	4.0
100	6.9	100	9.4	100	3.8
150	7.8	150	9.1	150	12.2
200	5.3	200	4.9	200	10.2
-200	15.2	-200	4.0	-200	65.9

The following is a typical analysis of the concentrate produced:

ANALYSIS OF SILVER CONCENTRATE

Constituent	Per cent	Constituent	Per cent
Silver.....	955 (oz.)	Zinc.....	0.6
Lead.....	2.6	Sulphur.....	34.06
Copper.....	11.8	Bismuth.....	0.1
Insoluble.....	3.8	Antimony.....	10.4
Iron.....	25.7	Arsenic.....	1.53

Treatment of Silver Ores. Cobalt.—A very complete history and description of the mines, mills and metallurgical treatment of the Cobalt, Ont., silver ores has been written by Messrs. J. J. Denny, Fraser D. Reed and R. H. Hutchinson and published in *Ont. Dept. Mines, 21st Ann. Rept.*, 1922. Today none of these mills is in operation.

MANGANESE-SILVER ORES

Oxidized silver ores containing the higher oxides of manganese are generally refractory to metallurgical treatment. Manganese fouls mercury if amalgamation of the gold content is attempted. A refractory compound of manganese and silver is formed,

probably a manganite, which is insoluble in cyanide solution and other common solvents for silver.

Caron Process.—The Caron Process (U. S. Patent 1232216, Aug. 3, 1917), described by G. H. Clevenger and M. H. Caron in *Bul. 226 U. S. B. of M.*, is based on the following principle: When oxidized ores containing a refractory compound of manganese and silver are heated in a reducing atmosphere, the higher manganese oxides are reduced to manganous oxides; and if cooled so as to prevent reoxidation, the refractory compound is rendered amenable to cyanidation. Refractory compounds of silver also can be so treated.

Manganese-silver ores occur generally in acid-eruptive rocks, chiefly rhyolite and dacite flows of later Tertiary age. Potassium-aluminum silicate is a vein material, and the vein quartz is changed from calcite. The manganese oxide is generally of secondary origin and is formed by atmospheric agencies. For the foregoing reason manganiferous ore from near the surface may be refractory but from depth may be amenable to treatment. "Wad," a hydrous manganese manganate, is common in the zone of oxidation.

Various treatments of the raw ore have proved unsuitable—concentration (including flotation), magnetic separation, chloridizing, roasting, volatilization, sulphuric acid and heating with organic matter. The Ag to Mn ratio persists in all sieve sizes from plus 20 to minus 200 mesh.

Laboratory tests were made in the United States and in Sumatra, followed by plant-scale runs in the latter country and a 50-ton plant at Pachuca, Mexico. Direct cyanidation of raw ore containing 2 to 10 per cent MnO_2 gave 50 per cent extraction of the silver, but ore with 25 per cent MnO_2 gave only 25 per cent extraction. The Caron process, on the other hand, extracted 92 per cent of the gold and 90 per cent of the silver. The pilot plant in Mexico successfully treated ore containing 2.8 to 13 per cent MnO_2 and 12 to 20 oz. silver. The Clevenger kiln (U. S. Patent 137908, May 24, 1921) was fired with producer gas with the following analysis: CO , 15 per cent; CH_4 , 5.5 per cent; H_2 , 4.6 per cent; CO_2 , 6 per cent, the remainder being nitrogen.

The general conclusions as to the operation and efficacy of the Caron process are:

1. Size of ore fed to the rotary kiln may be as coarse as 1 to 2 in.
2. Producer gas of 150 B.t.u. or higher, made from any fuel, may be used.
3. Temperature range is 500 to 700, best at 600°C.
4. Calcine should be discharged into an inert atmosphere or directly into cyanide solution.
5. Unaltered MnO₂ after calcining should be determined.
6. Alkalinity control is important.
7. Excess air should be used during grinding and agitation.
8. Gold in some manganese-silver ores is amenable to direct cyanidation, and its extraction is not increased by reduction. Gold in other ores follows the silver and is more or less refractory and is benefited as much as silver by reduction.
9. Silver extraction of 60 per cent from raw ore may be increased to from 88 to 96 per cent.
10. Cyanide consumption is not affected by MnO₂.
11. Lime consumption may be 16 lb. per ton.
12. A daily economic minimum-treatment plant for these ores is 200 tons, and the cost could approach \$1 (4 shillings) per ton, the equivalent of dead roasting and treating a low-sulphur ore.

Caron described the application of the process at Tambang Sawah, Sumatra, Dutch East Indies, a translation appearing in the *M.J.* (London) Feb. 19, 1927. The ore, mainly quartz, carried 18 per cent manganese dioxide, 30 oz. silver and 8 dwt. gold. After being crushed to 1 in., it was heated 4 hr. in a rotary kiln. It remained 1½ hr. in the reducing section of the kiln in producer gas at 600°C. The MnO₂ was reduced to 2 per cent. The ore was then ground and cyanided, yielding 87 per cent of the silver and 97 per cent of the gold, as compared to 25 per cent by raw treatment. Chemical consumption was 2.2 lb. cyanide, 5.5 lb. lime and 1.3 lb. zinc per ton ore.

A very complete bibliography covering the treatment of manganese-silver ores is given in *Bul. 226, U. S. B. of M.*, 1925, by Clevenger and Caron.

McClusky Process.—Manganese is found in varying percentages in the ores at Fresnillo, Mexico; although, fortunately, the average content is not enough to necessitate special treatment, ores from some parts of the mine contain sufficient manganese to affect seriously the extraction of the silver. To improve the extraction on this relatively small quantity of refractory ore, S. P. McClusky, formerly metallurgist with the Fresnillo company, developed a modified method of what has become known as the sulphur dioxide process for manganiferous silver ores.

This is described by W. E. Crawford in *Trans. 112, A.I.M.E.*, 1934.

Briefly, the method consists of (1) grinding of ore in water; (2) subjecting the pulp to the action of sulphur dioxide gas to dissolve the manganese minerals; (3) precipitating the dissolved manganese with a lime emulsion; (4) aerating the pulp and finally (5) cyaniding in the usual manner. The ideas involved in the method are (*a*) that part of the silver is in too close association with the manganese minerals which inhibit the action of cyanide solution on this silver; (*b*) that when these minerals are dissolved by SO₂ the associated silver is liberated and thereby becomes accessible to the solvent action of the cyanide solution. Moreover, if the dissolved manganese is then precipitated by lime emulsion and oxidized to the manganic state by aeration, it no longer affects extraction, although it still remains in the pulp.

The content of silver in the ore and the gain in extraction by sulphur dioxide treatment, plus the cost of the process, are the criteria by which the applicability of the process to manganiferous silver ores may be judged. At Fresnillo argentite is the predominant silver mineral. It is associated with pyrite and manganese mineral.

In practice, the ore is ground in a weak, "spent" solution of 0.008 per cent KCN so that 30 or 35 per cent passes through a 200-mesh sieve, and 1 per cent is coarser than 10 mesh. The pulp then flows to a 4-in. Wilfley centrifugal pump, which delivers the pulp to the top of the first and second of the sulphur-dioxide treatment towers, which are sealed, airtight chambers, three in number. These towers are of wood, 3 by 3 ft. in cross section and 17 ft. high, with wooden baffles lined with white-iron plates. The purpose of the baffle plates is to disperse the pulp as it falls through the tower, so that it may come into intimate contact with the ascending current of SO₂ gas. The flow of the pulp and SO₂ gas is countercurrent, the pulp is constantly enriched in acidity and the gas mixture progressively depleted of SO₂ from unit to unit. The final result is that the gas exhausting to the atmosphere contains slightly more than 1 per cent SO₂ indicating a total absorption of 85 per cent of the available SO₂. The gas used in this process and in the cyanide-regeneration

plant described elsewhere is produced by roasting the pyrite from flotation in seven-hearth Herreshoff furnaces.

From the SO₂ absorption towers the pulp passes through five conditioner tanks arranged in series. An emulsion of lime is added to the fourth tank for the purpose of precipitating the dissolved manganous and ferrous compounds, as manganous and ferrous hydrates. The low-pressure air in this and the fifth tank assists in oxidizing the manganous and ferrous compounds to manganic and ferric compounds. After passing through the last conditioner tank the pulp is returned to the mill for regrinding in a 6- by 14-ft. Traylor ball mill in closed circuit with a Dorr bowl duplex classifier. The overflow of this classifier, which averages 60 per cent minus 200-mesh material, joins the feed of the plant treating the regular silver ore.

The gain in extraction accomplished by the sulphur dioxide treatment varies considerably with different ores, but it appears to be in direct proportion to the amount of manganese dissolved by the gas, approximately 7 grams silver for every 0.1 per cent dissolved manganese. An increased recovery by this treatment, of 25 grams silver per ton, represented a substantial economic advantage when silver was quoted at around 30 cts. (U. S. A. currency) per oz.

The laboratory pilot test, carried out daily in conjunction with the plant treatment, often showed as much as 35 grams additional recovery of silver. Mixing of the SO₂-treated slimes with the general mill slimes made it difficult to check the actual additional recovery in the plant.

An interesting point is noted in connection with tests for the oxygen content of solution in the pulp leaving the final treatment tank of this unit. This solution is entirely devoid of free oxygen; moreover, it required several hours of vigorous agitation with air to satisfy the oxygen-consuming requirement and to render it susceptible to the absorption of free oxygen. In view of this, it is quite possible that a separate cyanide circuit for these treated slimes would be a distinct advantage, especially if it were so designed that several hours of agitation and aeration could be given prior to the addition of cyanide.

CHAPTER XIII

TREATMENT OF OLD RESIDUES

Source of Residues.—Residues may consist of untreated sand and slime from a stamp mill; sand and slime from a cyanide plant, following either raw treatment or a roasting process; raw-treated or roasted concentrates or flotation plant tailing. All of these types of residues require careful sampling and testing for treatment.

Many millions of tons of abandoned sand and slime have been treated both profitably and unprofitably in Arizona, California, Nevada, Montana, Mexico, Australia, Africa and India, and men are searching constantly for other dumps.

Condition of Gold.—Frequently much of the gold or silver in residues that have come from cyanide plants is in soluble form. It had either not been thoroughly washed, or the cyanide remaining in the residue may have continued the dissolving action for a time after the pulp was sent to the dump. The cyanidation of such material is sometimes largely a matter of washing out the dissolved metals and precipitating them as quickly and as cheaply as possible.

As proof of the soluble form of the gold and silver in dumps and of the continued dissolution, we have only to refer to the metal-bearing efflorescence that rises by capillary action to the surface of certain residue heaps. The dumps at Tonopah (silver) and at the Kalgurli and United Eastern (gold) are examples of this phenomenon; the material collected by scraping and sweeping is worth shipment to a smelter or feeding into a mill circuit.

At Tonopah, Nev., according to M. W. von Bernewitz in *M. and S.P.*, Jan. 18, 1913, the salts on the residue ponds formed a crust with the top $\frac{1}{8}$ in. of slime. The crust was scraped off with garden hoes or whisk brooms. Weather conditions were important; better results were obtained on warm days. The value of the product depended upon the care taken in scraping. The scrapings assayed \$30 to \$130 per ton and averaged \$40.

If the deposit was thick, one man could collect a ton of the material, although the production, generally, was but half this amount. The Tonopah companies let this job to lessees who paid as much as 30 per cent royalty. One lessee sent his product to the Selby smelter, California. At another mine, the scrapings were sampled, fed into a tube mill and mixed with the pulp being

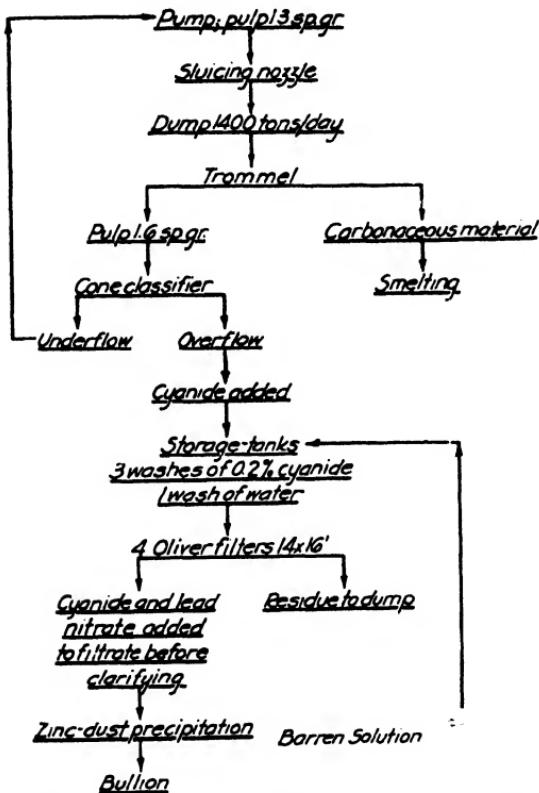


FIG. 130.—Flow-sheet of Golden Horseshoe tailings plant, Western Australia.

treated. At another plant, the scrapings after being sampled were mixed with the ore being fed to the mill.

At the United Eastern plant, according to E. M. Bagley, previously cited, the slime residue was distributed to ponds or benches. Each was built up 8 to 12 ft. and allowed to stand untouched for several months. Meanwhile, a layer of salts had accumulated on the surface. Three "crops" could be gathered before the material became too poor to be profitable. After

that, the residue was built up to a similar height, and the process repeated. The skimmings were trucked to the plant and fed to the ball-pebble mills. An analysis of this material follows.

ANALYSIS OF DUMP EFFLORESCENCE, PER CENT

Constituent	Amount	Constituent	Amount
Water.....	13.81	Sodium chloride.....	11.10
Silica.....	47.75	Iron.....	Trace
Calcium carbonate.....	23.47	Thiocyanates.....	Trace
Sodium carbonate.....	0.37	Cyanides.....	None
Sodium bicarbonate.....	2.81	Alumina.....	Trace
Sodium sulphate.....	4.60		

It cost 1.4 cts. per ton to treat this material as described.

AMERICAN RETREATMENT

Residue from Amalgamation.—1. Tailings at the Yellow Aster mine at Randsburg, southern California, consist of two dumps—one of 1,000,000 tons averaging \$1.35 per ton, and the other of 1,000,000 tons assaying 85 cts. The average assay for both dumps is \$1.04. The material is clean and consists of a highly siliceous gangue, according to C. R. Forbes in *E. and M.J.*, March, 1934. The ore had been stamp crushed through 20-mesh screens so that the slime content is low. The climate is favorable, and water and electric power are available.

The main problem at this plant was the handling of the tailings in and out of the leaching tanks, but this difficulty was overcome by the patented method of G. H. Wyman, Jr. The vats instead of being round are rectangular in shape; they are made of $\frac{3}{8}$ -in. steel and are 100 ft. long, 14 ft. wide and 6 ft. deep. In operation a scraper slides on the unloading rails and discharges the residue over the end of the tank into a hopper which in turn discharges on to a belt conveyor which carries the residue to a high dump. Loading of the vats is done by a scraper of 4 cu. yd. capacity which is operated by a three-drum electric hoist connected to a 75-hp. motor. The scraper carrier rope, bridle line and snatch-block arrangement is a modification of regular practice. A charge of 160 tons sand fills the vat to within 16 in. of the top. Nine hundred tons can be dragged 300 ft. in 4 hr.

A cycle of operations comprises $2\frac{1}{2}$ hr. each for loading and unloading, with three men. Cyanide solution is pumped through the bottom of the vats to within 6 in. of the top and remains there 24 hr., is then circulated for 2 hr. and stands another 24 hr., after which the solution percolates through the sand. Washing on the top continues for 2 days followed by a water wash, for a total period of 4 days. After the solutions are filtered, the gold is precipitated on zinc shavings.

Lime is added to the sand during loading operations at the rate of 3 lb. per ton. The solution carries 2.2 lb. cyanide per ton.



FIG. 131.—A typical Australian residue-treatment plant at Bendigo, Victoria.

Lead acetate is added, before washing starts, at the rate of 1 lb. per tank of sand. The zinc boxes are cleaned up after three tanks of sand have been treated, and the resulting precipitate is melted in a crucible using a gasoline burner, giving a low-grade bullion which is shipped to the mint. Slag from the melts is ground in a small crusher and concentrated on a laboratory-size Wilfley table. The concentrates are remelted, and the tails are saved and later sent to a smelter for treatment.

Operating costs are 27 cts.; and as the 70 per cent extraction was equivalent to 55 cts. gold, half of this is profit.

2. Tailings from the stamp-amalgamation-concentrator mills of the Mother Lode of California have been undergoing cyanide treatment for many years. This material comprises both accu-

mulations from earlier operations and tailings currently produced. Interest in tailing treatment is revived by the 1000-ton plant erected by the Kennedy Company to handle a dump of $1\frac{1}{2}$ million tons. The customer plant of the Amador Metals

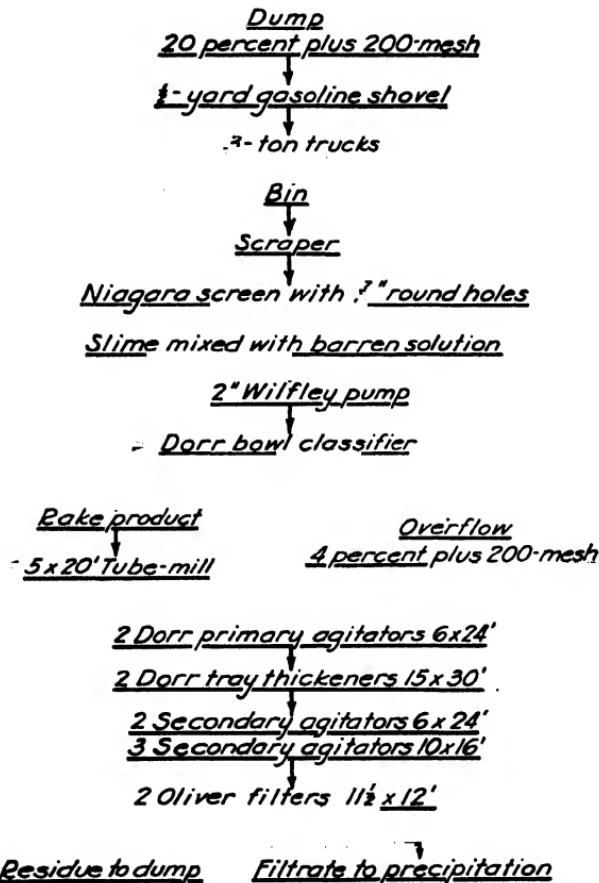


Fig. 132.—Retreatment plant at Teck-Hughes mine, Ontario.

Reduction Company receives 250 tons tailings a day from the Argonaut mill, and the Central Tailings Works receives 125 tons from the Central Eureka mill. According to M. N. Colman in *E. and M.J.*, November, 1933, a typical form of procedure is to separate all sand from slime by means of mechanical classifiers, leach the sand, agitate the thickened slime and filter it and precipitate the gold from the cyanide solutions on zinc dust.

Carbonaceous matter, generally associated with the slime, is troublesome. A coal-tar product is mixed with the slime and prevents premature precipitation. The investigation of E. S. Leaver and J. A. Woolf published in *T.P. 481, U. S. B. of M.*, 1930, will be remembered in this connection. They concluded that the gold lost in the mill tailings is largely in the remaining sulphides and carbonaceous material. The sand usually gives satisfactory recovery by direct cyanidation, but Leaver and Woolf prefer to float the mineral from the slime and roast and cyanide the concentrate. Evidently, some operators have followed this advice in California, and flotation of old tailings is being practiced.

Two-hundred-mesh Residue.--During suitable weather the Teck-Hughes company at Kirkland Lake, Ontario, treats old residue from previous cyanide operations at a rate of 225 tons per day. This material, of which 20 per cent remains on 200 mesh, contains 20 per cent moisture. Figure 132 shows the flow-sheet of operations.

The scraper mentioned in the flow-sheet is a mine drag scraper. The purpose of the screen is to eliminate miscellaneous rubbish which has become mixed with the old residue. The barren solution (1 lb. KCN and 1.2 lb. CaO per ton solution) comes from the plant in which ore is being milled. Barren solution and water are used as washes on the filters. The strongest solution uses 1 to 3 lb. KCN and 2 lb. CaO per ton. The pregnant solution is sent to the plant treating ore for precipitation.

The plant is more elaborate than many such installations. The 70 per cent recovery of gold is made at a cost of 95 cts. per ton residue handled.

MEXICAN RETREATMENT

Lixiviated Patio Tailings.--Near Fresnillo, Zacatecas, Mexico, the Fresnillo Company has treated several hundred thousand tons of tailing assaying 96.5 dwt. in silver and 0.32 dwt. in gold from a lixiviation plant which treated old patio tailings. After being screened of rock and plus $\frac{1}{2}$ -in. material, nearly 10 per cent remains on 4 mesh, and only 18 per cent is minus 100 mesh, according to W. E. Crawford in *Trans. 112, A.I.M.E.*, 1934. Apart from the low value of the material, the big problem was its transportation 1.2 miles against a head of 147 ft.

Considerable experimentation with sand pumps was necessary. Eventually, a 6-in. pipe line was laid with four 5-in. motor-driven sand pumps in series, each discharging into the suction of the next pump. Shell liners and impellers are of white iron and last 250 hr., pumping 550 to 650 tons dry sand and slime daily. Pulp dilution with mill solution is 4 to 1. The pulp is reground in a Traylor ball mill in closed circuit with an 8- by 26-ft. Dorr duplex classifier. Then it mixes with other ore pulp in the Fresnillo plant and is eventually separated into sand and slime.

CHAPTER XIV

COSTS AND POWER CONSUMPTION

ANALYSIS OF COSTS

The tabulated costs and power consumption in this chapter are from various sources but mostly from the descriptive articles in current publications. Data with regard to the cost of producing gold during the years 1934-1935 are from *T.P. 709, A.I.M.E.*, "Effect of Revaluation on the Gold-mining Industry," by John J. Croston, and are reproduced here by special permission. Only where a group of mines operates in a single district are costs comparable, and then only with reservations. In general, cost systems are fairly uniform, yet in studying costs of a number of plants it is noticeable that some details do not include coarse crushing or the shipment and outside treatment of concentrates. These operations are proper charges against ore dressing and treatment, from the first stage of coarse crushing, whether it be underground or on the surface, to the disposal of the residue and the recovery of bullion and returns from products sold; everything is properly chargeable against the mill. This includes the cost for labor, power, supplies, repairs and compensation.

As to power consumption, its distribution is simple. A motor is direct or belt connected to a mill or other machine, and a certain amount of power is metered. In California, Montana and Nevada in the United States and in India, New Zealand, Ontario and Transvaal, hydro- and steam-electric power are sold at fixed rates per horsepower year, and this simplifies charges.

COST OF PLANT CONSTRUCTION

Factors in the cost of building ore-dressing and treatment plants are type of ore, daily tonnage, accessibility of the mine and the simplicity or complexity of the process. A mill in which amalgamation alone is employed may cost \$450 to \$700 per ton ore treated per day; all-slime cyanide plants, \$800 to \$1400 per ton; all-flotation plants, \$600 to \$800; cyanide-flotation or

PRODUCTION AND COST DATA OF THE PRINCIPAL GOLD-MINING COMPANIES

Company	Year	Tonnage treated during year	Recovery, oz. per ton	Gold produced during year, oz.	Operating costs per ton before any charges or taxes*	Production costs per ounce before any charges or taxes*	Total costs per ounce after all charges and taxes*
South Africa							
Witwatersrand field 35 companies—totals and average.....	1934	40,055,750	0.2575	10,314,764	\$ 4.77	\$18.52	\$24.84
Government G.M. areas	Dec. 31, 1934	2,482,000	0.3690	915,851	4.36	11.82	24.92
New State areas	Dec. 31, 1934	1,122,000	0.3928	550,011	12.77	28.82	
Sub-Nigel.....	June 30, 1935	568,200	0.7919	449,977	8.45	10.67	18.91
Springs.....	Dec. 31, 1934	1,077,700	0.3813	421,688	4.86	12.42	24.19
Brakpan.....	Dec. 31, 1934	1,479,000	0.2734	404,411	4.86	17.76	25.61
East Geduld.....	Dec. 31, 1934	972,500	0.3409	331,492	4.51	13.24	21.52
Daggafontein.....	Dec. 31, 1934	959,900	0.3275	323,925	5.28	15.65	23.17
West Springs.....	Dec. 31, 1934	1,104,200	0.1571	173,314	3.76	23.95	27.89
Crown.....	Dec. 31, 1934	3,558,000	0.2815	1,001,618	4.86	17.26	24.62
Randfontein.....	Dec. 31, 1934	3,791,000	0.2928	730,936	4.80	24.89	26.64
New Modder.....	June 30, 1935	2,175,000	0.2491	541,876	3.52	14.09	21.28
East Rand Proprietary.....	Dec. 31, 1934	2,102,500	0.2272	477,712	5.30	23.35	26.30
West Rand Consolidated.....	Dec. 31, 1934	1,430,000	0.2371	388,388	4.24	17.85	24.09
Geduld Proprietary.....	Dec. 31, 1934	1,116,000	0.2804	324,136	3.63	12.60	22.93
Consolidated Main Reef.....	June 30, 1935	1,340,100	0.2193	293,911	5.32	24.25	28.29
Robinson Deep.....	Dec. 31, 1934	1,200,000	0.2508	301,017	4.69	18.69	24.75
Simmer and Jack.....	Dec. 31, 1934	1,073,900	0.2283	245,164	5.25	22.98	25.78
Modder East.....	June 30, 1935	1,074,000	0.2188	235,078	4.80	21.95	28.91
City Deep.....	Dec. 31, 1934	1,204,000	0.1962	236,210	5.29	26.95	28.43
Van Ryn Deep.....	Dec. 31, 1934	944,000	0.2334	220,416	4.73	20.25	25.69
Nourse.....	June 30, 1935	868,300	0.2254	195,746	5.84	25.89	30.76
Langlaistic.....	Dec. 31, 1934	1,007,000	0.1962	197,538	4.71	24.51	
Modder Deep P.....	Dec. 31, 1934	568,500	0.3387	182,537	3.74	11.06	18.39
Modder "H".....	Dec. 31, 1934	1,088,000	0.1809	123,335	3.61	18.89	23.90
Geldenhuys.....	Dec. 31, 1934	996,900	0.1628	161,301	4.27	26.21	29.65
Witwatersrand ("Knight").....	Dec. 31, 1934	958,000	0.1584	151,793	4.68	29.51	31.91
Durban Roodepoort Deep.....	Dec. 31, 1934	626,000	0.2424	151,772	6.00	24.76	26.65
Rose Deep.....	Dec. 31, 1934	764,500	0.1800	137,583	4.98	27.66	
Witwatersrand Deep.....	Dec. 31, 1934	590,850	0.1671	116,442	5.21	26.42	28.88

New Kleinfontein	Dec. 31, 1934	649,600	0.1751	113,707	5.47	31,24	32.04
Luijgaard's Vie.	June 30, 1935	609,900	0.2192	111,801	5.08	23,19	30.11
Van Ryn Gold	June 30, 1935	68,500	0.1445	4,09	5.02	31,89	31.89
Transvaal Gold	Mar. 31, 1935	243,400	0.2285	65,603	5.02	21,97	28.35
Glynn-Lydenburg	July 31, 1935	94,700	0.3228	33,410	7.80	22,12	29.73
Nigel Gold	Dec. 31, 1934	66,550	0.3112	20,711	6.82	21,92	21.92
Outside North America and South Africa*							
(Arranged by mining districts within continents)							
Africa:							
Ashanti, and Aboso	Sept. 30, 1935	192,110	1.0388	199,650	\$ 7.87	\$ 7.57	\$14.16
Taquish and Aboso	Mar. 31, 1935	149,687	0.3347	50,109	5.06	15.12	21.15
Ararion	Sept. 30, 1935	84,325	0.4063	34,261	8.76	21,56	24.57
Cam and Motor	June 30, 1935	308,400	0.3344	109,300 ^b	4.61	13,01	20.31
Globe and Phoenix	Dec. 31, 1934	72,180	0.8299	59,863	8.94	10,77	18.80
Wanderer	June 30, 1935	204,800	0.1763	36,113	3.52	25.73	25.73
Sherwood-Starr	June 30, 1935	78,600	0.2179	17,916 ^d	6.65	24.81	35.62
Lonely Reef	June 30, 1934	146,860 ^c	0.1067	19,024 ^c	3.23	27,04	29.83
Resende	Dec. 31, 1934	77,600	0.2359	18,307	4.57	19,36	24.15
Marcos	July 31, 1935	9 ^e		13,661	19,67	28,04	28,04
Gabait	Jan. 31, 1935	7,517	0.5589	4,202	11,82	32,21	32,21
Revere Dredging	June 30, 1935	1,260,000 ^c	22,54 ^c	8,259	1,14 ^e	17,03	21.77
Cie. Equatoriale Des Mines	Dec. 31, 1934	" ^w	" ^w	20,056	" ^w	17,35	31.07 ^e
South and Central America:							
St. John del Rey	Dec. 31, 1934	240,500	0.4027	96,857	8.31	23,67	27.65
Frontino	June 30, 1935	0.6830	49,927	16,25 ^b	23,67	31.11	31.11
New Goldfields of Venezuela	June 30, 1935	160,135	0.3699	49,620	7.88	25,43	30.78
Botanano	Dec. 31, 1934	23,979	0.8276	19,851	19,46	26,51	26.49
Venezuela	Dec. 31, 1934	13,775	1.1283	14,970	16,48	14,96 ^e	25.85 ^e
Asia:							
Nundydroog	Dec. 31, 1934	218,514 ^p	0.496	110,536 ^e	7.18 ^e	14,19	21.82
Myoore	Dec. 31, 1934	177,033	0.519	92,566	9.58	18,34	24.45
Champion Reef	Dec. 31, 1934	134,770 ^c	0.462	66,400 ^c	9.36 ^e	19,00	24.47
Oregum	Dec. 31, 1934	146,750 ^c	0.2576	50,303 ^c	6.13 ^e	25,57	29.08
Oriental Consolidated	Dec. 31, 1934	183,873	0.2243	41,251	3.50	15,60	29.74
Chosen	Mar. 31, 1935	134,245	0.2855	38,323	3.00	10,51	21.69
Australia, etc.:							
Lake View and Star	June 30, 1935	524,510 ^w	0.2490	132,241	5.27 ^e	20,49	25.38
Wiluna	Mar. 31, 1935	439,395	0.2465	108,319	3.83	15.54	29.56
Sons of Gwalia	Dec. 31, 1934	136,548	0.3130	42,740	5.66	18,10	22.96
North Kalgoorlie	Dec. 31, 1934	50,383	0.4697	23,215	7.97	17,29	24.59
Blackwater	Dec. 31, 1934	31,862	0.4697	14,966	6.82	14,51	25.40
Baltoor	Dec. 31, 1934	245,128	0.5473	134,161	5.41	9,88	14.68
Benguet	Dec. 31, 1934	250,553	0.4527	113,428	6.14	13,55	16.98
Bulolo Gold dredging	May 31, 1935	9,920,700 ^w	44.92 ^e	127,901	81.94 ^e	6.96	10.14

PRODUCTION AND COST DATA OF THE PRINCIPAL GOLD-MINING COMPANIES.—(Continued)

Company	Year	Tonnage treated during year	Recovery, or, per ton	Gold produced during year, oz.	Operating costs per ton before any charges or taxes*	Production costs per ounce after all charges and taxes*	Total costs per ounce
America and Canada							
America:							
Homestake.....	Dec. 31, 1934	1,440,662	0.3274	471,749	\$ 3.50	\$10.68	\$20.90
Alaska-Juneau.....	Dec. 31, 1934	4,302,600 ^b	0.0298	128,015	0.56	18.82 ^a	21.31 ^a
Natomas.....	Dec. 31, 1934	18,443,253 ^c	11.27 ^{c,d}	59,437	4.96 ^{e,f}	15.42	23.75
Yuba.....	Feb. 28, 1935	80,237	0.5541	67,096	17.77	26.96	
Idaho-Maryland.....	Dec. 31, 1934	68,590	0.3117	43,656	8.34	15.33	21.78
Argonaut.....	Sept. 30, 1935	234,056	0.0723	21,357	6.69	21.45	22.75
Carson Hill.....	Dec. 31, 1935 ^a	59,224	0.0745	16,980	2.22	30.75	32.13
Dairy Farm.....				4,412	1.65	22.10	
Canada:							
Hollinger.....	Dec. 31, 1934	1,900,480	0.2285	434,257	3.91	17.11	21.36
McIntyre-Porcupine.....	Mar. 31, 1935	862,100	0.281	242,236	4.19	14.91	20.82
Dome.....	Dec. 31, 1934	542,600	0.3723	206,163 ^a	3.88	10.30	15.39
Conlaurum.....	Dec. 31, 1934	138,114	0.2055	28,380	5.54	26.97	35.58
Buffalo-Ankerite.....	Dec. 31, 1934	131,718	0.1557	20,503	3.64	23.34	30.51
Anglo-Huronian "Vipond"	Dec. 31, 1935	104,764	0.1276	13,367	4.62 ⁱ	36.22 ⁱ	
Lake Shore.....	June 30, 1935	833,084	0.5526	460,442	5.50	9.95	17.25
Wright-Hargreaves.....	Aug. 31, 1935	350,196	0.6044	211,674	6.37	10.54	
Teck-Hughes.....	Aug. 31, 1935	382,958 ^k	0.3640	144,384 ^{m,n}	5.62 ^m	14.95	19.48
Sylvania.....	Mar. 31, 1935	124,936	0.405	50,601	5.93	14.66	24.30
Macassa.....	Mar. 31, 1935	66,534	0.4545	30,260	7.45	16.39	22.28
Tohorn.....	Dec. 31, 1934	36,230	0.563	20,401	9.89	17.57	28.27
Kirkland Lake Gold.....	Dec. 31, 1934	64,932	0.3128	20,316	7.63	24.39	27.19
Howey-Hollinger.....	Dec. 31, 1934	33,445	0.1316	4,409	6.00	45.00	
Ashley.....	Dec. 31, 1934	386,109 ^{pp}	0.1160	45,985	2.15	18.49	26.11
Central Patricia.....	Dec. 31, 1934	43,532	0.5524	13,181	8.69	28.68	36.72
Parkhill.....	Sept. 30, 1935	11,536 ^{qq}	6.373	11.28	20.43	28.03	
San Antonio.....	Dec. 31, 1934	20,100	0.4622	9,290	13.39	28.97	32.67
Central Manitoba.....	Aug. 31, 1935	64,294	0.3365	21,638	5.49	16.30	23.08
Siscoe.....	Dec. 31, 1934	58,068 ^r	0.2661	14,922	8.67	32.59	42.16
						63.394	10.89
						0.5106	
						124.131	

Beattie.....	Dec. 31, 1934	359,200	0.1473	52,905	3.72	25.23	31.67
Granada.....	Dec. 31, 1934	35,424	0.1937	6,864	7.46	38.48	47.65
Pioneer.....	Mar. 31, 1935	130,545	0.6846	87,030 ^{**}	5.42	8.14	18.38
Bratbine.....	Dec. 31, 1934	98,664	0.4591	45,596	4.44 ^{**}	9.67 ^{**}	18.39
Premier.....	Dec. 31, 1934	154,693	0.248 ^{**}	38,371	4.52
Reno.....	June 30, 1935	33,943	0.6050	20,538	9.95	16.44	22.74
Cariboo.....	Dec. 31, 1934	28,772	0.3828	11,014	9.68	25.28	32.42

* All conversations made with sterling at \$4.90; Australian pounds at \$3.90 and Canadian dollars at par. Costs per ton are for mined ore except where noted. Costs per ounce are based on total gold output including recoveries from tailing treatment, if any. The column on recovery refers to mined ore, and is not necessarily based on total tonnage milled.

Operating costs include usual mining and treatment charges and current development expense. In addition refining, mint and marketing charges and local administration costs and general charges, where it is possible to segregate, depreciation, depletion, when charged, directors' regular fees, but not extra compensation or bonuses; and all forms of taxes except taxes on dividends. In the various countries there are income, royalty, government lease participation, gold export duties, excess profits and other taxes. Where gold is required to be sold to local governments at a price below the world market, this constitutes an effect on tax on production, and has been added to the costs per ounce.

All costs based on sterling at \$4.90. The first eight companies have

properties leased from the government and subject to royalties and in some cases additional taxes.

From start in July, 1934. Leased but no lease payment, taxes or

phantis charges during first year.

^a Includes 2582 oz. from slags.

^b Includes 782 oz. from slags.

^c Plus 96,400 tons of tailings.

^d Includes 3345 oz. from tailings treatment.

^e Production mostly alluvial, some lode.

^f Cubic yards.

^g Per cubic yard with gold at \$35.

^h Includes depreciation.

ⁱ \$13.26 after crediting copper and silver.

^j Two-year period ending Dec. 31, 1935.

^k Tons trammed to mill.

^l \$20,39 crediting silver and lead.

^m Cubic yards.

ⁿ Per cubic yard with gold at \$35.

^o Per cubic yard.

^p Includes 2268 oz. from retreatment.

^q Mine expense only; no general or administrative expense.

^r Before taxes.

^s Plus 40,290 tons of tailings.

^{**} Includes proportionate tailings; mine ore alone estimated at \$5.17.

^{**} From start of operations May 27, 1934.

^{**} Period 16 months to Aug. 31, 1935.

^{**} Includes 267 oz. from slags.

^{**} Executive of development cost.

^{**} Includes 4,246 oz. silver.

^{**} Extremely high silver makes cost after crediting silver content not comparable.

^{**} Hydraulicking.

flotation-cyanide plants, at least \$1000 per ton-day. On the Rand, a plant consisting of crushers, stamps, tube mills and separate sand and slime treatment to handle 50,000 tons per month costs around £400,000 (\$1,900,000), whereas one with crushers, tube mills and all-slime treatment costs £300,000 (\$1,400,000). Those in charge of small mines being developed or already equipped to handle up to 100 tons ore daily will find considerable information of value in *I.C. 6800, U. S. B. of M.*, 1934, "Mining and Milling Practices at Small Gold Mines," by E. D. Gardner and C. H. Johnson.

TREATMENT COSTS

Amalgamation.—The Porcupine United Gold Mines at Timmins, Ontario, in 1931 had a 25-ton pilot mill for the amalgamation of a high-grade ore. Costs were as follows:

COST OF MILLING AND AMALGAMATING

Item	Labor	Power	Supplies	Total, cents per ton
Crushing.....	21.4	13.5	3.5	38.4
Grinding.....	42.9	18.0	16.2	77.1
Classifying, screening, conveying, refining.....	42.9	9.0	10.7	62.6
Miscellaneous.....	7.0	7.0
Total.....	107.2	40.5	37.4	185.1

Grinding, 65 per cent through 200 mesh.

The initial cost of building a flotation plant is usually considerably less than the initial cost of a cyanide plant of similar capacity. Likewise, the operating cost of flotation is usually lower than that of cyanidation. The capital cost of increasing the Benguet Consolidated plant in the Philippines from 500 to 750 tons per day with flotation was only a little more than one-fourth of the cost of increasing the all-cyanide plant from 300 to 500 tons.

GOLD

At Alaska-Juneau, Alaska.—The following milling costs at the concentrator of the preceding mill for the year 1931 are taken from an article by W. P. Scott, mill superintendent, in *E. and M.J.*, September, 1932.

The total operating cost per ton rejected by sorting follows:

	Cents per Ton
Coarse crushing per ton discarded.....	2.62
Sorting.....	3.27
Coarse screening.....	0.06
Waste-rock disposal.....	6.55
General operation and maintenance.....	1.71
	<hr/>
	14.21
Total tons discarded, 1931.....	1,863,332
	<hr/>
Cost of Fine Milling per Ton Milled	
Coarse crushing.....	2.62
Sorting.....	1.38
Screening.....	0.07
Intermediate crushing.....	0.72
Fine grinding.....	15.56
Concentrating.....	1.62
Retreatment.....	1.47
General operating and maintenance.....	4.16
	<hr/>
Total per ton.....	27.60
Tons milled.....	2,299,018

Average combined cost for treatment of total ore mined, (4,162,350 tons),
21.60 cts. per ton.

Grinding, 25 per cent through 200 mesh, through 10 mesh.

At Coniaurum, Ontario.—During 1933, the Coniaurum mill in the Porcupine district treated 145,657 tons ore at the following cost:

COST OF CONIAURUM TREATMENT	
Operation	Cents per Ton
Crushing, screening and conveying.....	24.9
Milling and classification.....	17.5
Agitation and thickening.....	5.9
Classification and precipitation.....	3.9
Cyanide and lime.....	8.4
Filtration.....	9.3
Residue disposal.....	5.4
Pumping pulp and solution.....	3.5
Refining.....	1.2
	<hr/>
Total.....	80.0

Grinding, 70 per cent through 200 mesh.

At Golden Cycle.—The Golden Cycle mill at Colorado Springs treats Cripple Creek and other ores at the following cost for the former class, at a rate of 818 tons per day:

COSTS (1935) IN THE GOLDEN CYCLE MILL

Operation	Cents per ton	Operation	Cents per ton
Unloading ore.....	7.10	Chemicals (cyanide 15½ cts.).....	20.23
Sampling (includes crushers and rolls).....	13.90	Supervision.....	3.92
Symons crusher.....	6.34	Office.....	4.16
Comminuters.....	11.57	Yard.....	1.75
Screens and elevators.....	2.06	Bullion.....	3.55
Roasters.....	63.51	Outside lighting.....	0.16
Chilean mills.....	12.49	By-product.....	1.13
Conveying.....	13.13	Flue dust.....	1.07
Amalgamating.....	2.23	Umpire expense.....	0.20
Classifying.....	0.83	Sundry buildings.....	1.39
Sand leaching.....	4.75	Railroad tracks.....	0.71
Slime treatment.....	5.85	Waterworks line.....	1.18
Filters.....	3.91	Track scales.....	0.03
Clarifier.....	1.10	Watchmen and fire.....	1.14
Precipitation (zinc).....	6.98	All insurance.....	3.36
Refining.....	4.19	General expense.....	5.85
Residue disposal.....	6.83		
Assaying.....	4.79	Total.....	\$2.214

Grinding, 35 per cent minus 200 mesh, all through 20 mesh.

Of the total, labor cost 54.5 cts. for operation and 10.9 cts. for repairs. The labor costs were distributed as follows: Roasting, 22 per cent; conveying, 11 per cent; sampling, 8 per cent; general items, 14 per cent; the remainder being fairly uniformly distributed in the other items. Power accounted for 25.5 cts., supplies 62.3 cts., other repair items 9.9 cts., and miscellaneous charges 31.1 cents per ton. Water cost almost 11 cts. per 1000 gallons.

At Hollinger, Ontario.—During 1935 the Hollinger plant treated 1,837,153 tons ore at a total cost of 64.7 cts. per ton, according to a statement sent direct to the author. These figures differ from the year 1934 in that the production was 63,337 tons less and the cost 5.2 cts. per ton more.

Some details of these costs will be more informative than the totals. For instance, general charges include 0.37 ct. for water; milling includes 7.29 cts. for rod mills, 5.90 cts. for pebble mills, 1.07 cts. for classifiers and for other charges around the circuit, 0.94 ct. Three-step decantation costs 2.52 cts., and primary filtration 2.36 cts. per ton treated.

COST OF HOLLINGER TREATMENT FOR 1935

Operation	Cents per ton	Operation	Cents per ton
General charges	3 03	Refinery	1 36
Crushing	6 16	Lime	1 55
Tramming and shoveling	2 42	Cyanide	6 95
Milling (grinding)	15 28	Assaying	1 44
Primary thickening	0 51	Heating	1 05
Agitating	0 70	Lighting	0 79
Pumping pulp	1 36	Superintendence and misc	1 14
Tabling	3 40	Sampling and experimental	0 89
Sulphide treatment	3 29	Building maintenance and	
3-step decantation	0 92	alteration	2 76
Primary filtration	1 49	Electrical equipment and	
Secondary filtration	2 53	maintenance	1 12
Tailing disposal	2 05		—
Pumping solution	0 68	Total	64 74
Precipitation	1 87		

This figure is 0.9 cts higher than the average of 2.5 cts for the past 7 years and this is due to major alterations in this section during the year.

Grinding 2 to 3 per cent plus 48 mesh and 65 per cent through 200 mesh.

At Homestake, South Dakota.—In treating about 1,400,000 tons ore a year at the Homestake mine, South Dakota, by a fairly simple process of amalgamation and cyanidation of the sand and slime of an ore carrying free gold and gold in pyrite, pyrrhotite and arsenopyrite, costs are substantially as follows:

COSTS (1929) OF MILLING AND TREATMENT AT HOMESTAKE	
Item	Cents per Ton
Crushing	3 8
Milling	27 6
Cyaniding	18 9
Total	50 3

AT SOUTH PLANT (MILLING ONLY)

Stamps	8 9
Rod mills	8 4
Amalgamating	1 4
Tube mills	1 4
Assaying and refining	1 3
Superintendence	1 1
Miscellaneous	1 0

Total

23 5

Grinding, 47 per cent through 200 mesh, through 50 mesh

At Howey, Ontario.—At Red Lake the Howey mill is treating 1100 tons ore daily by cyaniding a pulp of which 40 per cent through 200 mesh is fine enough to liberate the gold. The cost of doing this is as follows:

COST OF HOWEY TREATMENT	
Operation	Cents per Ton
Sorting.....	4.3
Crushing and conveying.....	11.1
Grinding and classifying.....	27.7
Thickening and agitating.....	4.0
Filtering.....	5.2
Clarifying and precipitating.....	2.2
Reagents.....	5.8
Laboratory and assaying.....	1.2
Refining.....	1.4
Heating.....	1.2
Residue disposal.....	1.6
Mill alterations.....	0.1
Pumping.....	3.8
 Total.....	 69.6

Grinding, 40 per cent through 200 mesh.

The original table in *C.M.J.*, October, 1934, did not include the first two items, so these have been added from the same article.

Costs at Kolar, India.—In 1933 the cost of ore treatment at Kolar by stamping, blanketing, amalgamation, grinding and cyaniding of slime was as follows, based on 659,448 tons ore:

TREATMENT COST AT KOLAR

Operation	Pence per ton	Cents per ton
Transport.....	4.4	8.8
Sorting and crushing.....	3.2	6.4
Stamping.....	14.4	28.8
Cone classification.....	2.6	5.2
Tube milling.....	10.6	21.2
Sand (old residue) and slime treatment.....	17.3	34.6
 Total.....	 52.5	 105.0

Grinding, 75 to 80 per cent through 200 mesh.

In 10 years, costs have been reduced 12 pence or 24 cts. per ton, a worth-while achievement when treating more than 600,000 tons in one year.

LAKE SHORE MILL COSTS

The following operating costs cover the new flow-sheet (see description of Lake Shore mill, Chap. XII) straight cyanidation when grinding to about 9 per cent plus 40 microns with about 38 per cent minus 10 microns. (44 microns = 325 mesh, and 9 microns = 1600 mesh.)

DETAILED OPERATING COSTS FOR FEBRUARY, MARCH AND APRIL, 1936*
(Daily Average Feed, 2500 Tons)

	Operation	Cost per ton of ore	Operation	Cost per ton of ore
1	Conveyor and ore bins	\$0.009		
2	Primary crushing	0.037		
3	Final crushing	0.065		
4	Niagara screens	0.006		
5	Mill conveyors	0.017		
6	Screen house (chip removal)	0.009		
7	Ball mills	0.163		
8	Simplex classifiers	0.010		
9	Elevation	0.013		
10	Tube mills	0.191		
11	Bowl classifiers	0.029		
12	Agitators	0.030		
13	Thickeners	0.044		
14	Chemicals	0.170		
15	Primary filters	0.054		
16	Final filters	0.072		
17	Clarification	0.008		
18	Precipitation	0.022		
19	Slimes disposal	0.010	Slimes disposal	0.010
20	Mill, general	0.066		0.067
21	Fire protection	0.001	General	
22	Refinery	0.024	Refining	0.024
23	Experimental	0.024	Experimental	0.024
	Total	\$1.074	Total	\$1.074

NOTE.—Assaying amounts to 2.2 cts. per ton. This has been charged against the units affected in the figures shown above.

* By R. W. Mancantelli.

AVERAGE RESULTS OF MILL ASSAYS FOR THE MONTHS OF JANUARY AND FEBRUARY, 1936,* LAKE SHORE
 (For the Purpose of Comparing the Commercial Results at 2300 Tons and 2500 Tons per Day)

Operation	January	February
Daily average tonnage.....	2320 tons	2500 tons
Grind { +40 microns.....	6.8	9.1
{ +28 microns.....	23.6	26.6
{ -10 microns.....	39.1	38.3
Heads.....	{ 0.5409 oz. \$11.181	0.5585 oz. \$11.544
Primary agitator feed.....	2.59	2.57
Primary agitator discharge.....	1.94	2.04
Secondary agitator feed.....	1.88	1.99
24 hours' agitation.....	0.44	0.484
37 hours' agitation.....	0.41	0.450
Secondary agitator discharge.....	0.404	0.445
Tertiary agitator feed.....	0.404	0.445
Tertiary agitator discharge.....	0.404	0.446
Final agitator feed.....	0.404	0.446
Final agitator discharge.....	0.404	0.446
Final tails.....	{ 0.404 0.0195 oz.	0.444 0.0215 oz.
Extraction.....	96.38%	96.15%
Sulphur.....	0.96%	0.97%
Cost per ton milled ^a	\$ 1.15	\$ 1.08

* Figures compiled by R. W. Mancantelli.

^a All costs include experimental work.

Costs at Lava Cap, Grass Valley, Calif.—When operating at one-third capacity, the 300-ton flotation plant of the Lava Cap Gold Mining Corporation, Grass Valley, Calif., treated gold-silver ore at a cost of 67.4 cts. per ton as follows:

Operation	COST (1935) OF FLOATING 100 TONS PER DAY	CENTS PER TON
Labor.....	18.5	
Compensation insurance.....	0.8	
Steel balls.....	11.7	
Chemical supplies.....	14.1	
Power.....	19.4	
Assaying.....	2.0	
Surface maintenance.....	0.9	
Total.....	67.4	

Grinding, 73 per cent through 100 mesh.

Concentrates are trucked to a smelter.

Costs at McIntyre-Porcupine, Ontario.—In the McIntyre-Porcupine mill 2400 tons daily is treated by flotation, producing 250 tons as a concentrate which is cyanided and tailing which is discarded. The cyanide plant has a capacity of 400 tons.

MILLING AND FLOATING IN MCINTYRE-PORCUPINE

Operation	Cents per Ton
Crushing and conveying.....	10.37
Flotation.....	27.64
Cyanidation.....	26.92
Refining.....	2.24
Assaying.....	1.43
Mill alterations.....	1.2
 Total.....	 69.8

Grinding, mill heads to 8 per cent plus 65 mesh, regrinding concentrate (250 tons) to 99 per cent minus 200 mesh. (1935 costs.)

Costs in Montana.—The concentrating plant of Montana Mines Corporation, near Helena, Mont., in 1931 reported the following costs to the U. S. Bureau of Mines:

MILLING AND FLOATING IN MONTANA

Operation	Cents per Ton
Crushing and conveying.....	16.2
Grinding.....	29.0
Flotation (reagents 6.3 cts.).....	25.8
Residue disposal.....	5.2
Water.....	1.2
Repairs and maintenance.....	23.3
 Total.....	 100.7

Grinding, 60 per cent through 200 mesh, all through 80 mesh.

This plant was treating 150 tons per day. Power, which cost 0.8 ct. per kw.-hr., amounted to 20.7 cts. per ton of the total charges.

At Morro Velho, Brazil.—In treating 264,694 tons gold-silver ore at the Morro Velho, Brazil, in 1932, by a seemingly complicated process of concentration and cyanidation of an ore carrying arsenopyrite and pyrrhotite, the cost was as follows, according to J. H. French and Harold Jones in *Bul., I.M. and M.*, July, 1933:

COST OF MILLING AND TREATMENT AT MORRO VELHO

Item	Pence per ton	Cents per ton
Hauling and breaking.....	4.261	8.522
Milling.....	4.747	9.494
Concentrating.....	1.318	2.636
Tube milling.....	5.491	10.982
Reconcentrating.....	0.754	1.508
Repairing and maintaining plant.....	3.208	6.416
Power and lighting.....	1.079	2.158
Sundries.....	0.331	0.662
Total milling and concentrating.....	21.189	42.378
Cyaniding and filtering (cyanide 24.5 pence or 49 cts.).....	29.162	58.324
Precipitating.....	1.538	3.076
Cleanup and acid treatment.....	1.026	2.052
Strakes.....	0.839	1.678
Repairing and maintaining plant.....	1.336	2.672
Power and lighting.....	0.553	1.106
Sundries.....	0.114	0.228
Special experimental work.....	0.852	1.704
Total cyanidation.....	35.420	70.840
Laboratory and assaying.....	2.225	4.450
Melting house.....	2.119	4.238
Refinery.....	0.404	0.808
Roasting furnaces.....	2.815	5.630
Tailing storage.....	0.551	1.102
Sundries.....	0.620	1.240
Administration and office.....	3.370	6.740
Total miscellaneous.....	12.104	24.208
Grand total.....	68.713	137.426

Grinding, 65 per cent through 200 mesh.

Cost of Milling Mother Lode Ore.—During 15 years the Argonaut Mining Company, California, milled more than 1,000,000 tons of ore by stamping, amalgamating and concentrating. Costs in that period ranged from 66 cts. in 1916 to \$1.41

in 1923 and to 90 cts. in 1926—low, high and more or less normal years. The charges during 1926 are as follows:

Cost (1931) OF MILLING AT THE ARGONAUT		Cents per Ton
Item		
Labor.....	27	
Power and water.....	13	
Supplies.....	10	
Trucking.....	2	
Concentrates treatment.....	33	
Insurance.....	2	
Miscellaneous.....	3	
Total	90	

Grinding all through 28 mesh by stamp mills. Capacity 250 tons per day.

Thirteen men run the mill. Concentrates are treated under contract near by.

At Premier, British Columbia.—To mill and float 170,000 tons gold-silver ore at the Premier mine, British Columbia, it cost \$1.067 per ton, divided as follows:

COST (1931) AT PREMIER, BRITISH COLUMBIA

Operation	Cents per ton	Operation	Cents per ton
Primary breaking.....	9.2	Small tools.....	0.2
Secondary breaking.....	7.0	Motors and accessories.....	1.3
Primary grinding.....	33.7	Lighting.....	0.8
Secondary grinding.....	11.8	Heating.....	0.8
Classifying.....	1.0	Experimenting.....	0.9
Floating.....	12.1	Building repairs.....	0.6
Reagents.....	5.9	Supervision.....	6.4
Filtering.....	5.1	Total.....	\$1.067
Handling pulp.....	9.0		
Lubricating.....	0.9		

Grinding, 70 per cent through 200 mesh. Capacity 500 tons per day.

Operation cost 72.2 cts. of which 23.1 cts. was for labor, and repairs cost 34.5 cts. of which 11.8 cts. was for labor. Power cost 27.4 cts. per ton.

At Reno, British Columbia.—At the Reno mine, British Columbia, treatment costs on nearly 1000 tons per month were as follows:

COSTS AT RENO PLANT, BRITISH COLUMBIA

Operation	Cents per ton	Operation	Cents per ton
Ore sorting.....	13.0	Reagents.....	35.4
Crushing.....	19.8	Power and air.....	85.6
Grinding.....	63.1	General expense.....	9.7
Tabling.....	5.9	Refining and amalgamating	11.8
Settling.....	23.0	Superintendence.....	26.8
Blanketing.....	5.6	Assaying.....	17.4
Clarifying.....	5.3	Workmen's compensation..	2.8
Precipitating.....	18.2		
			\$ 3.548

Grinding, 56 per cent through 200 mesh.

At United Eastern, Arizona.—The United Eastern plant at Oatman, Ariz., treated 325 tons per day of gold ore suitable for simple cyanidation when finely crushed. Costs during eight years (1917–1924) averaged as follows, 732,528 tons being treated:

COSTS AT UNITED EASTERN

Item	Cents per ton	Item	Cents per ton
General.....	16.1	Precipitating.....	9.5
Lighting.....	0.5	Refining.....	7.4
Water.....	10.8	Sampling.....	0.3
Coarse crushing.....	5.5	Assaying.....	3.1
Coarse grinding.....	35.6	Retreating dump skimmings	1.4
Fine grinding.....	46.9	Mill heating.....	3.1
Cyaniding.....	52.3	Solution recovery.....	1.0
Residue disposal.....	2.5	Experimental.....	0.1
Clarifying solutions.....	3.3	Total.....	*1.994

Grinding, 82 per cent through 200 mesh.

Water is unusually, yet naturally high in cost in the Southwest. (The California Rand in southern California had a charge of 60 cts. per ton for water.) When compared with other current charges for crushing, grinding and cyaniding, United Eastern costs appear high. Grinding cost would have been lower if the crushers had reduced the ore to an inch or less instead of to 2 in.

At Wright-Hargreaves, Ontario.—The Wright-Hargreaves plant at Kirkland Lake has a daily capacity of 1000 tons of ore which averages \$13 to \$14 per ton gold and a trace of silver (gold at \$20.67 per oz.). Costs average for 1935 as follows:

COST OF TREATING ORE AT WRIGHT-HARGREAVES

Operation	Cents per ton	Operation	Cents per ton
Crushing.....	12.6	Light and heat.....	2.2
Ball milling.....	24.9	Superintendence.....	2.8
Tube milling.....	23.6	Repairs.....	0.6
Filtering.....	11.3	Flotation.....	12.0
Thickening.....	7.3	Tailings disposal.....	3.2
Agitation.....	4.7	Reagents.....	10.9
Clarifying and precipitation.....	3.2	Cleanup, spills and elevating	0.6
Assaying and sampling.....	4.5	Experiments and research..	0.8
Refining.....	2.8	Total.....	\$ 1.279

Grinding, all through 200 mesh, and 85.0 per cent minus 325 mesh.

On the Rand.—1. The costs at two Rand mines using stamps and tube mills are as follows: One plant treats 79,000 tons per month, rejecting 27 per cent as waste and grinding 77 per cent of the mill feed through 100 mesh. The other discards 23 per cent, all mine fines passing direct to tube mills, and 92 per cent is ground through 100 mesh.

COST OF TREATMENT AT TWO RAND MINES

Operation	Pence per ton	Cents per ton
Ore transport from shaft to plant.....	0.48	2.28
Crushing and sorting.....	6.11	5.10
Stamp milling.....	6.78	5.29
Tube milling.....	7.74	10.97
Cyaniding.....	10.93	9.29
Total.....	32.04	32.93
	64.08	65.86

Compare these items with those of the following example of tube milling only:

2. East Geduld, Witwatersrand, has one of the new all-sliming plants. Its two-stage crushers and three-stage tube mills and

cyanide plant have a monthly capacity of 86,000 tons. The cost of doing this work in August, 1934, was:

COST OF ALL-SLIMING CYANIDATION

Operation	Cost per ton	
	Pence	Cents
Sorting and crushing.....	6.5	13
Tube milling.....	13.5	27
Cyaniding.....	9.5	19
Total.....	29.5	59

These operations are described elsewhere in this book.

SILVER

At Presidio, Texas.—Cost details of crushing and treatment at the Presidio silver mine near Shafter, Tex., are as follows, based on 54,644 tons treated:

DIRECT COSTS (1929) AT PRESIDIO MILL Cents per Ton

Department	Operation		Repairs		Power	Total
	Labor	Supplies	Labor	Supplies		
Breaking.....	1.8	0.2	0.7	0.6	1.1	4.4
Intermediate breaking.....	1.7	0.5	1.0	4.4	2.0	9.6
Tube milling.....	6.7	9.4	2.5	9.7	14.8	43.1
Agitating.....	5.7	0.5	1.1	2.1	8.2	17.6
Filtering.....	2.2	0.2	1.4	3.2	2.6	9.6
Precipitating.....	1.8	9.7	0.7	1.8	3.0	17.0
Drying and melting.....	2.4	1.8	0.1	0.1	4.4
Pumping and elevating.....	0.2	1.2	0.4	2.2	4.2	8.2
Tabling.....	8.3	1.8	1.4	2.2	1.8	15.5
Experimenting.....	0.4	0.7	1.1
Chemicals.....	1.0	49.5	0.2	50.7
Supervision.....	18.6	18.6
Total.....	50.8	75.5	9.5	26.3	37.7	\$ 1.99

In British coinage this is equivalent to 99.5 pence.
Grinding, 75 per cent through 200 mesh.

The chemicals consisted of 10.78 lb. of 70 per cent available lime per ton, 2.37 lb. of sodium cyanide, and 0.84 oz. of zinc dust per oz. of silver in precipitate.

CONCENTRATES

COST OF CYANIDING CONCENTRATE AT McINTYRE-PORcupine
(Apr. 1, 1935, to Mar. 31, 1936)
Cents per Ton of Ore Milled

Operation	Cost	Operation	Cost
Tube milling.....	3.56	Precipitating.....	1.37
Agitating.....	0.94	Reagents.....	11.50
Classifying.....	0.44	Heating and lighting.....	0.37
Thickening.....	0.47	Supervision.....	1.98
Pumping and elevating.....	2.38		
Filtering.....	3.22	Total.....	26.92
Clarifying.....	0.69		

As the ratio of concentration is 9 or 10 to 1, the cost of concentrate treatment would approximate \$2.50 per ton concentrate.

OLD-RESIDUE TREATMENT

At Bendigo, Australia.—Simple leaching of relatively clean stamp-mill tailings is being done at a cost of 38 to 41 pence or 76 to 82 cts. per ton, at a rate of 2500 tons per day.

At Yellow Aster, California.—Simple leaching of stamp-mill tailings in 160-ton vats costs as follows:

LEACHING COST AT YELLOW ASTER	
Item	Cents per Ton
Cyanide at 16½ cts. per ton.....	2.3
Lime at \$12 per ton.....	1.9
Water at \$1.25 per 1000 gal.....	4.3
Zinc shavings at 18½ cts. per lb.....	1.3
Lead acetate.....	0.1
Labor.....	12.6
Power.....	2.6
Sundry supplies.....	2.1
 Total, excluding royalty.....	 27.2

At Kalgoorlie, Western Australia.—The sluicing and washing of old bromocyanided tailings at the former Horse Shoe mine costs 27 pence or 55 cts. per ton at a rate of 1400 tons per day.

Dump Treatment in India.—Following is the cost of dump treatment at Kolar, based on 499,000 tons ground and cyanided in 1933:

COST OF TREATMENT IN INDIA

Operation	Pence per ton	Cents per ton
Transport and vortex mixing.....	5.7	11.4
Cone classification.....	1.8	3.6
Tube milling.....	6.5	13.0
Slime treatment.....	16.2	32.4
Total.....	30.2	60.4

COST OF LEACHING COARSE ORE

The Mountain Copper Company, California, is cyaniding by leaching a gossan, and the Wasp No. 2, in South Dakota, leached a quartzite at the following cost:

COST OF LEACHING IN 1931 AND 1932

Item	Gossan	Quartz- ite
	Cents per ton	Cents per ton
Labor.....	18.2	41.1
Cyanide.....	18.8
Zinc.....	3.1
Material.....	17.6	
Power.....	3.1	11.6
Repairs.....	8.6
Local charges.....	4.3	
Total.....	43.2	83.2

The respective daily tonnages are 700 and 105. Under "Gossan," material covers cyanide, lime and zinc. Under "Quartzite," power means fuel.

POWER

At Alaska-Juneau.—Power consumption at Alaska-Juneau is as follows, according to W. P. Scott in *E. and M.J.*, September, 1932:

POWER CONSUMPTION AT JUNEAU

Operation	Kilo-watt-hours	Operation	Kilo-watt-hours
Coarse crushing.....	0.4806	Tube mills.....	8.4848
Sorting.....	0.2586	Ball mill (tube-mill floor).....	5.7266
Coarse-waste disposal.....	0.2268	Concentrating.....	0.2088
Screening (Niagara).....	0.0130	Retreatment.....	0.2088
Intermediate crushing (Symons Cone).....	0.6036	General mill operating.....	0.9104
Screening (Hum-mer).....	0.1991	Assaying and sampling.....	0.0230
Ball mills.....	9.5295	Total on rock trammed.....	6.2200
Rolls.....	2.9771		

At Golden Cycle, Colorado.—The Golden Cycle mill, in treating 818 tons per day, consumed 27.6 kw.-hr. per ton with the following distribution:

POWER DISTRIBUTION AT GOLDEN CYCLE

Operation	Percentage Total Power
Sampling, including jaw crushers and rolls.....	5.56
Symons crusher and comminutors.....	18.59
Roasting.....	17.96
Chilean mills.....	24.37
Classifying, screening, elevating, conveying.....	14.33
Amalgamating.....	0.47
Cyaniding:	
Sand treatment.....	1.40
Slime treatment:	
Agitating and settling.....	5.88
Filtering.....	1.96
Aerating.....	2.32
Precipitating.....	0.36
Refining.....	0.52
Tailing disposal.....	6.34
By-product.....	0.94
Total.....	100.00

Power charge was 0.96 ct. per kw.-hr.

At McIntyre-Porcupine.—Power distribution in the McIntyre-Porcupine plant, which crushes 2100 tons per day and cyanides only about one-tenth of it, is as follows:

POWER DISTRIBUTION AT THE MCINTYRE-PORCUPINE	
Operation	Horsepower
Crushing (excluding underground).....	279
Flotation (including five tube mills).....	1516
Cyaniding (including two tube mills).....	558
Refinery.....	17
Assay office.....	70
Water supply.....	33
Heating and lighting.....	80
Total.....	2553

At Premier, British Columbia.—In crushing, grinding and floating 170,000 tons ore at the Premier, British Columbia, the distribution of 5,304,450 kw.-hr. was as follows:

POWER DISTRIBUTION AT PREMIER

Operation	Per cent	Operation	Per cent
Primary breaking.....	4.46	Floating.....	19.58
Secondary breaking.....	7.47	Filtering.....	4.02
Primary grinding.....	22.84	Pumping.....	18.70
Secondary grinding.....	18.32	Lighting.....	3.17
Classifying.....	1.44	Total.....	100.00

At United Eastern.—The United Eastern all-sliming cyanidation plant at Oatman, Ariz., used the following amount of power per ton of ore treated:

POWER USED IN UNITED EASTERN
Kilowatt-hours per Ton

Operation	Quantity	Operation	Quantity
Lighting.....	0.090	Clarifying.....	0.317
Coarse crushing.....	0.445	Precipitating.....	0.701
Coarse grinding.....	6.709	Refining.....	0.001
Fine grinding.....	8.597	Solution on recovery.....	0.116
Cyaniding.....	6.045	Total.....	23.021

The coarse crushing from 2 in. was done in ball mills, which is much larger feed than they get nowadays. This is reflected in the power for the ball mills and ball-pebble mills.

At Wright-Hargreaves.—The installed horsepower in the Wright-Hargreaves crushing and treatment plant is 2577, this being equivalent to 2.6 hp. per ton milled.

On the Rand.—1. The percentage of power consumed in different departments of three Rand mills is as follows:

PERCENTAGE OF POWER CONSUMPTION ON RAND

Operation	Plant		
	1	2	3
Crushers.....	3.51	3.60	5.19
Stamps.....	12.11	16.10
Tube mills.....	73.69	56.94	55.86
Pumps.....	4.96	13.07	7.54
Cyanidation.....	17.81	14.25	15.27

2. At East Geduld, the present consumption of power is as follows:

MILL OUTPUT AND POWER USED

Tube mills	Minus 200 mesh per horsepower-day, tons	Minus 200 mesh per tube-mill day, tons
Primary (3).....	0.598	149.87
Secondary (10).....	0.799	150.51
Tertiary (4).....	1.109	266.43
Average (17).....	0.841	178.55

The tube-mill power units consumed per ton minus 200-mesh material are 21.28 kw.-hr.

There are four sizes of tube mills running on the Rand— $5\frac{1}{2}$ by 22, 6 by 16, $6\frac{1}{2}$ by 20 and 8 by 16 ft. Motors attached to these are 100 to 175 hp. for the first size, 250 hp. for the third size and 350 hp. for the largest mill. The drive through reduction gear is common practice. The average power is 18 kw.-hr. per ton minus 100-mesh product.

CHAPTER XV

USEFUL REFERENCE INFORMATION

This concluding chapter contains miscellaneous information, methods of calculation, factors and other items to which the millman may refer to refresh his memory.

CALCULATION OF RECOVERY IN CONCENTRATE

This method is from E. M. Hamilton's *Manual of Cyanidation*, 1920:

Let C = assay value of concentrate.

H = assay value of heads.

T = assay value of tails.

R = ratio of concentration.

W = weight of concentrate in per cent.

P = percentage recovery.

L = percentage loss in tails.

$$R = \frac{C - T}{H - T}$$

$$W = \frac{H - T}{C - T} \times 100$$

$$P = \frac{100 \times C(H - T)}{H(C - T)}$$

$$P = \frac{C \times 100}{H \times R}$$

$$L = \frac{100 \times T(C - H)}{R(C - T)}$$

THE ELEMENTS

The table following has been compiled from the *Journal of the American Chemical Society*, 1933, and from the *Handbook of Chemistry and Physics*, 1933:

Name	Symbol	Atomic number	Atomic weight	Weight, lb per cu. ft.	Melting point, °C.
Actinium (rare).....	Ac	89	227		
Aluminum.....	Al	13	26.97	168.5	658.7
Antimony.....	Sb	51	121.7 ^a	388.3	630.0
Argon (a gas).....	A	18	39.944	80.4 (liquid)	-188
Arsenic.....	As	33	74.33	357.7	500
Barium.....	Ba	56	137.36	236.0	850
Beryllium.....	Be	4	9.02	114.9	1280
Bismuth.....	Pt	83	209.00	603.7	271
Boron.....	B	5	10.82	152.9	2400
Bromine (a liquid).....	Br	35	79.916	194.8	-7.3
Cadmium.....	Cd	48	112.41	539.9	320.9
Calcium.....	Ca	20	40.08	96.1	810
Carbon.....	C	6	12.00	140.5	3000
Cerium.....	Ce	58	140.13	430.7	700
Cesium (rare).....	Cs	55	132.81	116.9	26
Chlorine (a gas).....	Cl	17	35.457	94.1 (liquid)	-101.5
Chromium.....	Cr	24	52.01	432.0	1520
Cobalt.....	Co	27	58.94	543.7	1480
Columbium (rare).....	Cb	41	93.3	524.4	
Copper.....	Cu	29	63.57	518.1	1063
Dysprosium (rare).....	Dy	66	162.40		
Erbium (rare).....	Er	68	167.64	298.0	
Europium (rare).....	Eu	63	152.0		
Fluorine (a gas).....	F	9	19.00	71.2 (liquid)	-223
Gadolinium (rare).....	Gd	64	157.3		
Gallium.....	Ga	31	69.72	369.1	29.75
Germanium.....	Ge	32	72.60	340.9	958
Gold.....	Au	79	197.2	1204.8	1063
Hafnium (rare).....	Hf	72	178.0		
Helium (a gas).....	He	2	4.002	9.4 (liquid)	-271
Holmium (rare).....	Ho	67	163.5		
Hydrogen (a gas).....	H	1	1.0078	4.4 (liquid)	-250
Illinium (rare).....	Il	61	146		
Indium (rare).....	In	49	114.8	454.5	155
Iodine.....	I	53	126.92	308.4	113.5
Iridium.....	Ir	77	193.1	1399.6	2350(?)
Iron.....	Fe	26	55.84	438.9 to 493.2	1530
Krypton (a gas).....	Kr	36	83.7	134.8 (liquid)	-157
Lanthanum (rare).....	La	57	138.92	383.9	810(?)
Lead.....	Pb	82	207.22	687.0	327
Lithium (rare).....	Li	3	6.940	33.3	186
Lutecium (rare).....	Lu	71	175.0		
Magnesium.....	Mg	12	24.32	108.7	651
Manganese.....	Mn	25	54.93	463.2	1210
Masurium.....	Ma	43			
Mercury (liquid).....	Hg	80	220.61	845.6	-39.7

Name	Symbol	Atomic number	Atomic weight	Weight, lb. per cu. ft.	Melting point, °C.
Molybdenum.....	Mo	42	96.0	562.5	2410
Neodymium (rare).....	Nd	60	144.27	434.5	840(?)
Neon (a gas).....	Ne	10	20.183		-253(?)
Nickel.....	Ni	28	58.69	536.9	1452
Nitrogen (a gas).....	N	7	14.008	50.6 (liquid)	-211
Osmium.....	Os	76	190.8	1404.6	2700
Oxygen (a gas).....	O	8	16.000	71.2 (liquid)	-218
Palladium.....	Pd	46	106.7	759.1	1549
Phosphorus (soft).....	P	15	13.02	146.1	44.2
Platinum.....	Pt	78	195.23	1334.1	1755
Polonium (rare).....	Po	84	210.0		
Potassium (soft).....	K	19	39.10	54.3	62.3
Praseodymium (rare).....	Pr	59	140.92	404.2	940
Protoactinium (rare).....	Pa	91			
Radium (rare).....	Ra	88	225.97		700
Radon (an emanation).....	Rn	86	222.		
Rhenium (rare).....	Re	75	186.31		
Rhodium.....	Rh	45	102.91	776.6	1950
Rubidium (rare).....	Rb	37	85.44	95.6	38
Ruthenium (rare).....	Ru	44	101.7	752.9	2450(?)
Samarium (rare).....	Sm or Sm	62	150.43	480.7	1300
Scandium (rare).....	Sc	21	45.10		
Selenium.....	Se	34	79.2	268.4	217
Silicon.....	Si	14	28.00	140.7	1420
Silver.....	Ag	47	107.880	650.5	960.5
Sodium (soft).....	Na	11	22.997	60.6	97.5
Strontium.....	Sr	38	87.63	156.1	900(?)
Sulphur.....	S	16	32.06	124.9	106.8
Tantalum.....	Ta	73	181.4	1036.3	2900
Tellurium.....	Te	52	127.5	375.8	452
Terbium (rare).....	Tb	65	159.2		
Thallium (rare).....	Tl	81	204.39	740.4	302
Thorium.....	Th	90	232.12	705.4	1842
Thulium (rare).....	Tm	69	169.4		
Tin.....	Sn	50	118.70	455.1	231.9
Titanium.....	Ti	22	47.90	280.9	2000
Tungsten.....	W	74	184.0	1161.1	3400
Uranium.....	U	92	238.14	1167.4	1850
Vanadium.....	V	23	50.95	372.1	1720
Xenon (a gas).....	Xe	54	131.3	219.7 (liquid)	-140
Ytterbium (rare).....	Yb	70	173.5		
Yttrium (rare).....	Y	39	88.92	237.2	1490
Zinc.....	Zn	30	65.38	448.9	419.4
Zirconium.....	Zr	40	91.22	402.0	1700(?)

GENERAL CONVERSION FACTORS

One of the most frequent tasks of the engineer is the rapid and accurate conversion of the units of measure of one system into the different but related units of another system.

Each engineer remembers the conversion factors of those units that he uses most frequently. If, however, he has to convert units other than these, he often has to consult several handbooks before the desired conversion factor is found.

With the recognition of a need for a concise table of conversion factors, the following data applicable to metallurgical needs are taken from a compilation by Anthony Anable of The Dorr Company for its staff:

EXPLANATION

1. Data are arranged alphabetically.
2. Unless designated otherwise, the British measures of capacity are those used in the United States, and the units of weight and mass are avoirdupois units.
3. The word gallon, used in any conversion factor, designates the United States gallon. To convert into the Imperial gallon, multiply the United States gallon by 0.83267. Likewise, the word ton designates a short ton, 2000 lb.
4. The figures 10^{-1} , 10^{-2} , 10^{-3} and so on denote 0.1, 0.01, 0.001, respectively.
5. The figures 10^1 , 10^2 , 10^3 and so on denote 10, 100, 1000, respectively.
6. With respect to the properties of water, it freezes at 32°F. and is at its maximum density at 39.2°F. In the conversion factors given using the properties of water, calculations are based on water at 39.2°F. *in vacuo*, weighing 62.427 lb. per cu. ft., or 8345 lb. per United States gallon.
7. "Parts per Million," designated as p.p.m., is always by weight and is simply a more convenient method of expressing concentration, either dissolved or undissolved material. As a rule, p.p.m. is used where percentage would be so small as to necessitate several ciphers after the decimal point, as one part per million is equal to 0.0001 per cent.
8. As used in the sanitary field, p.p.m. represents the number of pounds of dry solids contained in 1,000,000 lb. of water, including solids. In this field, 1 p.p.m. may be expressed as 8.345 lb. of dry solids to 1,000,000 United States gallons of water. In the metric system, 1 p.p.m. may be expressed as 1 gram of dry solids to 1,000,000 grams of water, or 1 milligram per liter.
9. In arriving at parts per million by means of pounds per million gallons or milligrams per liter, it may be mentioned that the density of the solution or suspension has been neglected; and if this is appreciably different from unity, the results are slightly in error.

CONVERSION FACTORS

Multiply	By	To obtain
Acres	43,560	Square feet
Acres	4047	Square meters
Acres	1.562×10^{-3}	Square miles
Acres	4840	Square yards

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Acre-feet	43,560	Cubic feet
Acre-feet	325,851	Gallons
Acre-feet	1233.49	Cubic meters
Atmospheres	76.0	Centimeters of mercury
Atmospheres	29.92	Inches of mercury
Atmospheres	33.90	Feet of water
Atmospheres	10,333	Kilograms per square meter
Atmospheres	14.70	Pounds per square inch
Atmospheres	1.058	Tons per square foot
Barrel (oil) (cement)	42	Gallons
Bags or sacks (cement)	376	Pounds
Bags or sacks (cement)	94	Pounds
Board-feet	144 sq. in. × 1 in.	Cubic inches
British thermal units	0.2520	Kilogram-calories
British thermal units	777.5	Foot-pounds
British thermal units	3.927×10^{-4}	Horsepower-hours
British thermal units	107.5	Kilogram-meters
British thermal units	2.928×10^{-4}	Kilowatt-hours
B.t.u. per minute	12.96	Foot-pounds per second
B.t.u. per minute	0.02356	Horsepower
B.t.u. per minute	0.01757	Kilowatts
B.t.u. per minute	17.57	Watts
Centares (centiares)	1	Square meters
Centigrams	0.01	Grams
Centiliters	0.01	Liters
Centimeters	0.3937	Inches
Centimeters	0.01	Meters
Centimeters	10	Millimeters
Centimeters of mercury	0.01316	Atmospheres
Centimeters of mercury	0.4461	Feet of water
Centimeters of mercury	136.0	Kilograms per square meter
Centimeters of mercury	27.85	Pounds per square foot
Centimeters of mercury	0.1934	Pounds per square inch
Centimeters per second	1.969	Feet per minute
Centimeters per second	0.03281	Feet per second
Centimeters per second	0.036	Kilometers per hour

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Centimeters per second	0.6	Meters per minute
Centimeters per second	0.02237	Miles per hour
Centimeters per second	3.728×10^{-4}	Miles per minute
Centimeters per second per second	0.03281	Feet per second per second
Cubic centimeters	3.531×10^{-6}	Cubic feet
Cubic centimeters	6.102×10^{-2}	Cubic inches
Cubic centimeters	10^{-6}	Cubic meters
Cubic centimeters	1.308×10^{-6}	Cubic yards
Cubic centimeters	2.642×10^{-4}	Gallons
Cubic centimeters	10^{-3}	Liters
Cubic centimeters	2.113×10^{-3}	Pints (liquid)
Cubic centimeters	1.057×10^{-3}	Quarts (liquid)
Cubic feet	2.832×10^4	Cubic centimeters
Cubic feet	1728	Cubic inches
Cubic feet	0.02832	Cubic meters
Cubic feet	0.03704	Cubic yards
Cubic feet	7.48052	Gallons
Cubic feet	28.32	Liters
Cubic feet	59.84	Pints (liquid)
Cubic feet	29.92	Quarts (liquid)
Cubic feet per minute	472.0	Cubic centimeters per second
Cubic feet per minute	0.1247	Gallons per second
Cubic feet per minute	0.4720	Liters per second
Cubic feet per minute	62.43	Pounds of water per minute
Cubic feet per second	0.646317	Million gallons per day
Cubic feet per second	448.831	Gallons per minute
Cubic inches	16.39	Cubic centimeters
Cubic inches	5.787×10^{-4}	Cubic feet
Cubic inches	1.639×10^{-6}	Cubic meters
Cubic inches	2.143×10^{-6}	Cubic yards
Cubic inches	4.329×10^{-3}	Gallons
Cubic inches	1.639×10^{-2}	Liters
Cubic inches	0.03463	Pints (liquid)
Cubic inches	0.01732	Quarts (liquid)
Cubic meters	10^6	Cubic centimeters
Cubic meters	35.31	Cubic feet

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Cubic meters	61,023	Cubic inches
Cubic meters	1.308	Cubic yards
Cubic meters	264.2	Gallons
Cubic meters	10 ³	Liters
Cubic meters	2113	Pints (liquid)
Cubic meters	1057	Quarts (liquid)
Cubic yards	7.646×10^5	Cubic centimeters
Cubic yards	27	Cubic feet
Cubic yards	46,656	Cubic inches
Cubic yards	0.7646	Cubic meters
Cubic yards	202.0	Gallons
Cubic yards	764.6	Liters
Cubic yards	1616	Pints (liquid)
Cubic yards	807.9	Quarts (liquid)
Cubic yards per minute	0.45	Cubic feet per second
Cubic yards per minute	3.367	Gallons per second
Cubic yards per minute	12.74	Liters per second
Decigrams	0.1	Grams
Deciliters	0.1	Liters
Decimeters	0.1	Meters
Degrees (angle)	60	Minutes
Degrees (angle)	0.01745	Radians
Degrees (angle)	3600	Seconds
Degrees per second	0.01745	Radians per second
Degrees per second	0.1667	Revolutions per minute
Degrees per second	0.002778	Revolutions per second
Dekagrams	10	Grams
Dekaliters	10	Liters
Dekameters	10	Meters
Drams	27.34375	Grains
Drams	0.0625	Ounces
Drams	1.771845	Grams
Fathoms	6	Feet
Feet	30.48	Centimeters
Feet	12	Inches
Feet	0.3048	Meters
Feet	1/8	Yards
Feet of water	0.02950	Atmospheres

CONVERSION FACTORS —(Continued)

Multiply	By	To obtain
Feet of water	0 8826	Inches of mercury
Feet of water	304 8	Kilograms per square meter
Feet of water	62 43	Pounds per square foot
Feet of water	0 4335	Pounds per square inch
Feet per minute	0 5080	Centimeters per second
Feet per minute	0 01667	Feet per second
Feet per minute	0 01829	Kilometers per hour
Feet per minute	0 3048	Meters per minute
Feet per minute	0 01136	Miles per hour
Feet per second	30 48	Centimeters per second
Feet per second	1 097	Kilometers per hour
Feet per second	0 5921	Knots
Feet per second	18 29	Meters per minute
Feet per second	0 6818	Miles per hour
Feet per second	0 01136	Miles per minute
Feet per second per second	30 48	Centimeters per second per second
Feet per second per second	0 3048	Meters per second per second
Foot-pounds	$1 286 \times 10^{-4}$	British thermal units
Foot-pounds	$5 050 \times 10^{-7}$	Horsepower-hours
Foot-pounds	$3 241 \times 10^{-1}$	Kilogram-calories
Foot-pounds	0 1383	Kilogram-meters
Foot-pounds	$3 766 \times 10^{-1}$	Kilowatt-hours
Foot-pounds per minute	$1 286 \times 10^{-1}$	British thermal units per minute
Foot-pounds per minute	0 01667	Foot-pounds per second
Foot-pounds per minute	$3 030 \times 10^{-1}$	Horsepower
Foot-pounds per minute	$3 241 \times 10^{-4}$	Kilogram-calories per minute
Foot-pounds per minute	$2 260 \times 10^{-6}$	Kilowatts
Foot-pounds per second	$7 717 \times 10^{-2}$	British thermal units per minute
Foot-pounds per second	$1 818 \times 10^{-1}$	Horsepower
Foot-pounds per second	$1 945 \times 10^{-2}$	Kilogram-calories per minute
Foot-pounds per second	$1 356 \times 10^{-3}$	Kilowatts
Gallons	3785	Cubic centimeters
Gallons	0 1337	Cubic feet
Gallons	231	Cubic inches
Gallons	$3 785 \times 10^{-3}$	Cubic meters
Gallons	$4 951 \times 10^{-3}$	Cubic yards

CONVERSION FACTORS.—(Continued)		
Multiply	By	To obtain
Gallons	3.785	Liters
Gallons	8	Pints (liquid)
Gallons	4	Quarts (liquid)
Gallons, Imperial	1.20095	U. S. gallons
Gallons, U. S.	0.83267	Imperial gallons
Gallons water	8.3453	Pounds of water
Gallons per minute	2.228×10^{-3}	Cubic feet per second
Gallons per minute	0.06308	Liters per second
Gallons per minute	8.0208	Cubic feet per hour
Gallons per minute	8.0208	Overflow rate (feet per hour)
Area (square feet)		
Gallons water per minute	6.0086	Tons water per 24 hours
Grains (troy)	1	Grains (avoirdupois)
Grains (troy)	0.06480	Grams
Grains (troy)	0.01467	Pennyweights (troy)
Grains (troy)	2.0833×10^{-3}	Ounces (troy)
Grains	980.7	Dynes
Grains	15.43	Grains
Grams	10^{-3}	Kilograms
Grams	10^3	Milligrams
Grams	0.03527	Ounces
Grams	0.03215	Ounces (troy)
Grams	2.205×10^{-3}	Pounds
Grams per centimeter	5.600×10^{-3}	Pounds per inch
Grams per cubic centimeter	62.43	Pounds per cubic foot
Grams per cubic centimeter	0.03613	Pounds per cubic inch
Grams per liter	58.417	Grains per gallon
Grams per liter	8.345	Pounds per 1000 gallons
Grams per liter	0.062427	Pounds per cubic foot
Grams per liter	1000	Parts per million
Hectares	2.471	Ares
Hectares	1.076×10^5	Square feet
Hectograms	100	Grams
Hectoliters	100	Liters
Hectometers	100	Meters
Hectowatts	100	Watts
Horsepower	42.44	British thermal units per minute
Horsepower	33,000	Foot-pounds per minute

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Horsepower	550	Foot-pounds per second
Horsepower	1.014	Horsepower (metric)
Horsepower	10.70	Kilogram-calories per minute
Horsepower	0.7457	Kilowatts
Horsepower	745.7	Watts
Horsepower (boiler)	33,479	British thermal units per hour
Horsepower (boiler)	9.803	Kilowatts
Horsepower-hours	2547	British thermal units
Horsepower-hours	1.98×10^6	Foot-pounds
Horsepower-hours	641.7	Kilogram-calories
Horsepower-hours	2.737×10^5	Kilogram-meters
Horsepower-hours	0.7457	Kilowatt-hours
Inches	2.540	Centimeters
Inches of mercury	0.03342	Atmospheres
Inches of mercury	1.133	Feet of water
Inches of mercury	345.3	Kilograms per square meter
Inches of mercury	70.73	Pounds per square foot
Inches of mercury	0.4912	Pounds per square inch
Inches of water	0.002458	Atmospheres
Inches of water	0.07355	Inches of mercury
Inches of water	25.40	Kilograms per square meter
Inches of water	0.5781	Ounces per square inch
Inches of water	5.202	Pound per square foot
Inches of water	0.03613	Pound per square inch
Kilograms	980,665	Dynes
Kilograms	2.205	Pounds
Kilograms	1.102×10^{-3}	Tons (short)
Kilograms	10^3	Grams
Kilogram-calories	3.968	British thermal unit
Kilogram-calories	3086	Foot-pounds
Kilogram-calories	1.558×10^{-3}	Horsepower-hours
Kilogram-calories	1.162×10^{-3}	Kilowatt-hours
Kilogram-calories per minute	51.43	Foot-pounds per second
Kilogram-calories per minute	0.09351	Horsepower
Kilogram-calories per minute	0.06972	Kilowatts
Kilograms per meter	0.6720	Pound per foot
Kilograms per square meter	9.678×10^{-5}	Atmospheres
Kilograms per square meter	3.281×10^{-5}	Feet of water

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Kilograms per square meter	2.896×10^{-3}	Inches of mercury
Kilograms per square meter	0.2048	Pound per square foot
Kilograms per square meter	1.422×10^{-3}	Pound per square inch
Kilograms per square millimeter	10^6	Kilograms per square meter
Kiloliters	10^3	Liters
Kilometers	10^5	Centimeters
Kilometers	3281	Feet
Kilometers	10^3	Meters
Kilometers	0.6214	Miles
Kilometers	1094	Yards
Kilometers per hour	27.78	Centimeters per second
Kilometers per hour	54.68	Feet per minute
Kilometers per hour	0.9113	Feet per second
Kilometers per hour	0.5396	Knots
Kilometers per hour	16.67	Meters per minute
Kilometers per hour	0.6214	Miles per hour
Kilometers per hour per second	27.78	Centimeters per second per second
Kilometers per hour per second	0.9113	Feet per second per second
Kilometers per hour per second	0.2778	Meters per second per second
Kilowatts	56.92	British thermal units per minute
Kilowatts	4.425×10^4	Foot-pounds per minute
Kilowatts	737.6	Foot-pounds per second
Kilowatts	1.341	Horsepower
Kilowatts	14.34	Kilogram-calories per minute
Kilowatts	10^3	Watts
Kilowatt-hours	3415	British thermal units
Kilowatt-hours	2.655×10^6	Foot-pounds
Kilowatt-hours	1.341	Horsepower-hours
Kilowatt-hours	860.5	Kilogram-calories
Kilowatt-hours	3.671×10^6	Kilogram-meters
Liters	10^3	Cubic centimeters
Liters	0.03531	Cubic feet
Liters	61.02	Cubic inches
Liters	10^{-3}	Cubic meters
Liters	1.308×10^{-3}	Cubic yards
Liters	0.2642	Gallons
Liters	2.113	Pints (liquid)
Liters	1.057	Quarts (liquid)
Liters per minute	5.886×10^{-4}	Cubic feet per second
Liters per minute	4.403×10^{-3}	Gallons per second

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Lumber Width (in.) × thickness (in.)	12	
Meters	100	Centimeters
Meters	3.281	Feet
Meters	39.37	Inches
Meters	10^{-3}	Kilometers
Meters	10^3	Millimeters
Meters	1.094	Yards
Meters per minute	1.667	Centimeters per second
Meters per minute	3.281	Feet per minute
Meters per minute	0.05468	Feet per second
Meters per minute	0.06	Kilometers per hour
Meters per minute	0.03728	Miles per hour
Meters per second	196.8	Feet per minute
Meters per second	3.281	Feet per second
Meters per second	3.6	Kilometers per hour
Meters per second	0.06	Kilometers per minute
Meters per second	2.237	Miles per hour
Meters per second	0.03728	Miles per minute
Microns	10^{-6}	Meters
Miles	1.609×10^6	Centimeters
Miles	5280	Feet
Miles	1.609	Kilometers
Miles	1760	Yards
Miles per hour	44.70	Centimeters per second
Miles per hour	88	Feet per minute
Miles per hour	1.467	Feet per second
Miles per hour	1.609	Kilometers per hour
Miles per hour	0.8684	Knots
Miles per hour	26.82	Meters per minute
Miles per minute	2682	Centimeters per second
Miles per minute	88	Feet per second
Miles per minute	1.609	Kilometers per minute
Miles per minute	60	Miles per hour
Milliers	10^3	Kilograms
Milligrams	10^{-3}	Grams
Milliliters	10^{-3}	Liters
Millimeters	0.1	Centimeters
Millimeters	0.03937	Inches
Milligrams per liter	1	Parts per million
Million gallons per day	1.54723	Cubic feet per second
Miner's inch	1.5	Cubic feet per minute

CONVERS'ON FACTORS.—(Continued)

Multiply	By	To obtain
Minutes (angle)	2.909×10^{-4}	Radians
Ounces	16	Drams
Ounces	437.5	Grains
Ounces	0.0625	Pounds
Ounces	28.349527	Grams
Ounces	0.9115	Ounces (troy)
Ounces	2.790×10^{-5}	Tons (long)
Ounces	2.835×10^{-5}	Tons (metric)
Ounces, troy	480	Grains
Ounces, troy	20	Pennyweights (troy)
Ounces, troy	0.08333	Pounds (troy)
Ounces, troy	31.103481	Grams
Ounces, troy	1.09714	Ounces, avoirdupois
Ounces (fluid)	1.805	Cubic inches
Ounces (fluid)	0.02957	Liters
Ounces per square inch	0.0625	Pounds per square inch
Pennyweights (troy)	24	Grains
Pennyweights (troy)	1.55517	Grams
Pennyweights (troy)	0.05	Ounces (troy)
Pennyweights (troy)	4.1667×10^{-2}	Pounds (troy)
Pounds	16	Ounces
Pounds	256	Drams
Pounds	7000	Grains
Pounds	0.0005	Tons (short)
Pounds	453.5924	Grains
Pounds	1.21528	Pounds (troy)
Pounds	14.5833	Ounces (troy)
Pounds (troy)	5760	Grains
Pounds (troy)	240	Pennyweights (troy)
Pounds (troy)	12	Ounces (troy)
Pounds (troy)	373.24177	Grams
Pounds (troy)	0.822857	Pounds (avoirdupois)
Pounds (troy)	13.1657	Ounces (avoirdupois)
Pounds (troy)	3.6735×10^{-4}	Tons (long)
Pounds (troy)	4.1143×10^{-4}	Tons (short)
Pounds (troy)	3.7324×10^{-4}	Tons (metric)
Pounds of water	0.01602	Cubic feet
Pounds of water	27.68	Cubic inches
Pounds of water	0.1198	Gallons
Pounds of water per minute	2.670×10^{-4}	Cubic feet per second

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Pounds per cubic foot	0.01602	Grams per cubic centimeter
Pounds per cubic foot	16.02	Kilograms per cubic meter
Pounds per cubic foot	5.787×10^{-4}	Pounds per cubic inch
Pounds per cubic inch	27.68	Grams per cubic centimeter
Pounds per cubic inch	2.768×10^4	Kilograms per cubic meter
Pounds per cubic inch	1728	Pounds per cubic foot
Pounds per foot	1.488	Kilograms per meter
Pounds per inch	178.6	Grams per centimeter
Pounds per square foot	0.01602	Feet of water
Pounds per square foot	4.883	Kilograms per square meter
Pounds per square foot	6.945×10^{-3}	Pounds per square inch
Pounds per square inch	0.06804	Atmospheres
Pounds per square inch	2.307	Feet of water
Pounds per square inch	2.036	Inches of mercury
Pounds per square inch	703.1	Kilograms per square meter
Quadrants (angle)	90	Degrees
Quadrants (angle)	5400	Minutes
Quadrants (angle)	1.571	Radians
Quarts (dry)	67.20	Cubic inches
Quarts (liquid)	57.75	Cubic inches
Quintal:		
Argentine	101.28	Pounds
Brazil	129.54	Pounds
Castile, Peru	101.43	Pounds
Chile	101.41	Pounds
Mexico	101.47	Pounds
Metric	220.46	Pounds
Radians	57.30	Degrees
Radians	3438	Minutes
Radians	0.637	Quadrants
Radians per second	57.30	Degrees per second
Radians per second	0.1592	Revolutions per second
Radians per second	9.549	Revolutions per minute
Radians per second per second	573.0	Revolutions per minute per minute
Radians per second per second	0.1592	Revolutions per second per second
Revolutions	360	Degrees
Revolutions	4	Quadrants
Revolutions	6.283	Radians
Revolutions per minute	6	Degrees per second

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Revolutions per minute	0.1047	Radians per second
Revolutions per minute	0.01667	Revolutions per second
Revolutions per minute per minute	1.745×10^{-3}	Radians per second per second
Revolutions per minute per minute	2.778×10^{-4}	Revolutions per second per second
Revolutions per second	360	Degrees per second
Revolutions per second	6.283	Radians per second
Revolutions per second	60	Revolutions per minute
Revolutions per second per second	6.283	Radians per second per second
Revolutions per second per second	3600	Revolutions per minute per minute
Seconds (angle)	4.848×10^{-6}	Radians
Square centimeters	1.076×10^{-3}	Square feet
Square centimeters	0.1550	Square inches
Square centimeters	10^{-4}	Square meters
Square centimeters	100	Square millimeters
Square feet	2.296×10^{-5}	Acres
Square feet	929.0	Square centimeters
Square feet	144	Square inches
Square feet	0.09290	Square meters
Square feet	3.587×10^{-8}	Square miles
Square feet	$\frac{1}{6}$	Square yards
Square inches	6.452	Square centimeters
Square inches	6.944×10^{-3}	Square feet
Square inches	645.2	Square millimeters
Square kilometers	247.1	Acres
Square kilometers	10.76×10^6	Square feet
Square kilometers	10^6	Square meters
Square kilometers	0.3861	Square miles
Square kilometers	1.196×10^6	Square yards
Square meters	2.471×10^{-4}	Acres
Square meters	10.76	Square feet
Square meters	3.861×10^{-7}	Square miles
Square meters	1.196	Square yards
Square miles	640	Acres
Square miles	27.88×10^6	Square feet
Square miles	2.590	Square kilometers
Square miles	3.098×10^6	Square yards
Square millimeters	0.01	Square centimeters
Square millimeters	1.550×10^{-3}	Square inches

CONVERSION FACTORS.—(Continued)

Multiply	By	To obtain
Square yards	2.066×10^{-4}	Acres
Square yards	9	Square feet
Square yards	0.8361	Square meters
Square yards	3.228×10^{-7}	Square miles
Temperature (°C.) +273	1	Absolute temperature (°C.)
Temperature (°C.) +17.78	1.8	Temperature (°F.)
Temperature (°F.) +460	1	Absolute temperature (°F.)
Temperature (°F.) -32	$\frac{5}{9}$	Temperature (°C.)
Tons (long)	1016	Kilograms
Tons (long)	2240	Pounds
Tons (long)	1.12000	Tons (short)
Tons (metric)	10^3	Kilograms
Tons (metric)	2205	Pounds
Tons (short)	2000	Pounds
Tons (short)	32,000	Ounces
Tons (short)	907.18486	Kilograms
Tons (short)	2430.56	Pounds (troy)
Tons (short)	0.89287	Tons (long)
Tons (short)	29,166.66	Ounces (troy)
Tons (short)	0.90718	Tons (metric)
Tons of water per 24 hour	83.333	Pounds water per hour
Tons of water per 24 hour	0.16643	Gallons per minute
Tons of water per 24 hour	1.3349	Cubic feet per hour
Watts	0.05692	British thermal units per minute
Watts	44.26	Foot-pounds per minute
Watts	0.7376	Foot-pounds per second
Watts	1.341×10^{-3}	Horsepower
Watts	0.01434	Kilogram-calories per minute
Watts	10^{-3}	Kilowatts
Watt-hours	3.415	British thermal units
Watt-hours	2655	Foot-pounds
Watt-hours	1.341×10^{-3}	Horsepower-hours
Watt-hours	0.8605	Kilogram-calories
Watt-hours	367.1	Kilogram-meters
Watt-hours	10^{-3}	Kilowatt-hours
Yards	91.44	Centimeters
Yards	3	Feet
Yards	36	Inches
Yards	0.9144	Meters

METALS AND THEIR MINERALS BASED ON INFORMATION FROM DANA'S TEXTBOOK OF MINERALOGY

Metal	Composition	Mineral	Formula	Specific gravity	Hardness	Color and characteristic ^a
Antimony.....	Sulphide	Stibnite (antimony glance)	Sn ₂ S ₃	4.5 to 4.6	2.0	Lead-gray, metallic luster, slightly sectile
Cobalt.....	Arsenide	Smaltite	CoAs ₂	6.4 to 6.6	5.5 to 6	Tin-white, metallic, opaque
	Sulpharsenide	Cobaltite	CoAsS	6.0 to 6.3	5.5 to 6	Silver-white to reddish, cubic, metallic
	Metallic Oxide	Native	Cu	8.8 to 8.9	2.5 to 3	Copper-red, malleable and ductile
	Oxide	Melaconite (black oxide)	CuO	5.8 to 6.2	3 to 4	Black
	Carbonate	Cuprite (red oxide)	Cu ₂ O	5.8 to 6.1	3.5 to 4	Red, brittle
	Sulphide	Malachite	Cu ₂ CO ₃ .Cu(OH) ₂	3.9 to 4.0	3.5 to 4	Green, brittle
	Sulphide	Aurrite	2 Cu ₂ CO ₃ .Cu(OH) ₂	3.7 to 3.8	3.5 to 4	Aur-blue
	Sulphide	Chalcocyprite	Cu ₂ FeS ₃	4.1 to 4.3	3.5 to 4	Brass-yellow, brittle
	Sulphide	Bornite	Cu ₅ FeS ₄	4.9 to 5.4	3.0	Copper-red to brown, brittle
	Sulphide	Covellite	Cu ₁₁ S	4.59 to 5.5	2.5 to 3	Indigo-massive
	Sulphide	Chalcocite (copper glance)	Cu ₁₁ S	5.5 to 5.8	2.5 to 3	Lead-gray, often tarnished, sectile
	Antimonide	Tetrahedrite (gray-copper)	Cu ₁₂ As ₃ S ₃	4.4 to 5.1	3 to 4	Flint-gray to iron-black
	Sulpharsenate Silicate	Enargite	Cu ₁₂ SiO ₄ + 2H ₂ O	4.4 to 5.1	3 to 4	Gray-black, brittle
		Chrysocolla	Cu ₂ SiCH ₃ O ₃	2 to 2.2	2 to 4	Turquoise-blue, translucent, vitreous, rather sectile
	Oxychloride	Atacamite	Cu ₁₆ to 19.3	3 to 3.5	3 to 3.5	Bright, bottle green, brittle
	Metallic	Native	(Au,Ag)Te[Au:Ag = 1:1]	7.9 to 8.3	2.5 to 3	Gold-yellow, malleable and ductile
	Telluride	Sylvanite	7.9 to 8.3	1.5 to 2	Steel-gray to silver-white to yellow, brittle	
	Telluride	Calaverite	(Au,Ag)Te[Au:Ag = 6:1]	9.0 to 9.7	2.5 to 3	Pale bronze-yellow, massive
	Carbon	Peteite	(Ag,Au)Te[Au:Ag = 1:3]	2.0 to 2.23	1 to 2	Steel-gray to iron-black, brittle
	Graphite.....	Plumbago				Iron-black, darksteel gray, greasy, flexible
Iron.....	Metallic Oxide	Native	Fe	7.3 to 7.8	4 to 5	Steel-gray to iron black malleable
	Oxide	Hematite	Fe ₂ O ₃	4.9 to 5.3	5.5 to 6	Steel-gray, red
	Oxide	Limonite	2Fe ₂ O ₃ .3H ₂ O	3.6 to 4.0	5 to 5.5	Brown, opaque
	Carbone	Magnetite	Fe ₃ O ₄	5.16 to 5.16	5.5 to 6.5	Iron-black, magnetic
	Sulphide	Siderite	FeCO ₃	3.8 to 4.0	3.5 to 4	Gray-brown, reddish, brittle
	Sulphide	Pyrrhotite	FeS ₂	4.9 to 5.1	6 to 6.5	Brass-yellow, brittle
	Sulphide	Marcasite	FeS ₂	4.85 to 4.9	6 to 6.5	Pale bronze-yellow, brittle
	Sulphide	Pyrrhotite	FeS ₂ to Fe _{1-x} S _x	4.58 to 4.64	3.5 to 4.5	Bronze-yellow to copper-red, tarnished
	Sulpharsenide	Mispickel	FeAsS	5.9 to 6.2	5.5 to 6	Silver-white, brittle
	Titanate	Ilmenite	FeTiO ₃	4.5 to 5	5 to 6	Iron-black, slightly magnetic
	Wolframite	Wolframite	(Fe,Mn)WO ₄	7.2 to 7.5	5 to 5.5	Brown-black, brittle
	Chromate	Chromite	FeCr ₂ O ₄	4.3 to 4.57	5.5	Iron and brown black, brittle
	Manganate	Franklinite	(Fe,Zn,Mn)O(Fe,Mn) ₂ O ₃	5.1 to 5.22	5.5 to 6.5	Iron-black

Lead	Metallic Carbonate Sulphate Sulphide Metallic Sulphide Oxide Arsenide Sulphide	Pb PbCO ₃ PbSO ₄ PbS Hg HgS MoS ₂ MoO ₃ NiAs NiS	Native Cerussite Anglenite Galenite Native Chinnabar Molybdenite Molybdate Nicolite Kupfernickel Millerite	11.37 6.46 to 6.57 6.3 7.4 to 7.6 13.59 8.0 to 8.2 4.5 to 4.8 4.5 to 4.8 7.3 to 7.67.5 5.3 to 5.65.3	1.5 3 to 3.5 2.75 to 3 2.5 to 2.75 ... 2 to 2.5 1 to 1.5 1 to 1.5 to 5.5 to 3.5	Lead-gray White-gray, very brittle White, gray, green, very brittle White, gray, brilliant Tin-white, metallic, brilliant Cochineal-red, sectile Lead-gray, foliated, flexible, sectile Straw-yellow, capillary, earthy Pale copper-red, metallic, opaque Brass to bronze-yellow, capillary crystals
Silver	Silicate Metallic Antimonide Chloride	H ₂ (NiMg) ₂ SiO ₄ Ag ₂ Sh to Ag ₃ Sb AgCl[Cl 24.7, Ag 75.3] (Ag,Bz)	Garnierite Native Dysorite Ceratoxyrite (horn silver) Bromellite Argentite Sulphantimonide Sulpharsenide Sulphantimonide	2.3 to 2.8 10.1 to 11.1 9.4 to 9.8 5.5 to 6.1 5.8 to 6	2.5 to 3 2.5 to 3 3.5 to 4 1 to 1.5	Apple-green, amorphous, soft, friable Massive, fliform, malleable Silver-white, sectile Pear-gray, resembles wax, highly sectile
Tin	Sulphantimonide Oxide Sulphotannate	Ag ₃ S[12.9, Ag 87.1] Ag ₃ SbS ₃ Ag ₃ AsS ₃ Ag ₃ SbS ₃ Ag ₃ SnO ₃ Cu ₃ Fe ₂ SnS ₆ (Fe,Mn)WO ₄	Pyrrhotite (dark ruby silver) Proustite (light- ruby silver) Stephanite (brittle silver ore) Polybasite Cassiterite Stannite Tin pyrites Wolframite	7.2 to 7.36.2 5.7 to 5.8 5.5 to 5.6 6.2 to 6.3 6.0 to 6.2 6.8 to 7.1 4.3 to 4.5 7.2 to 7.5	2 to 2.5 2.5 to 3 2 to 2.5 2 to 3 6 to 7 4.0 to 5.5 5 to 5.5	Lead-gray, sectile Black, gray-black, brittle Scarlet-vermilion, brittle
Tungsten	Ferrous	Mn ₃ WO ₁₂ Ca ₃ WO ₁₂ WO ₃ ZnO	Föhnerite Scheelite Tungstate Zincite	5.9 to 6.1 5.4 to 5.7 4.3 to 4.45 ZnS	4.5 to 5 2 to 4.5 4 to 5.5 4.1 to 5	Red to hair-brown, nearly black White, yellow, brownish, translucent Yellow, greenish, earthy Deep red, orange-yellow, perfect cleavage
Zinc	Manganous Calcareous Oxide Oxide	Smithsonite Sphalerite Galamine Willemite	Smithsonite Sphalerite Galamine Willemite	3.9 to 4.1 3.4 to 3.5 3.4 to 3.5 3.89 to 4.18	3.5 to 4 4.5 to 5.0 4.5 to 5.0 5.5	Yellow, brown, black, perfect cleavage White, yellowish to brown, brittle White, apple-green, brown

Note.—The preceding table is from Hamilton's *Manual of Cyanidation*, 1920.

CONVERSION TABLE FOR WEIGHTS
(With authentic abbreviations according to International Critical Tables, Vol. I)

Weight	Grains	Penny-weights	Troy ounces	Avoirdupois ounces	Avoirdupois pounds	Grams
1 grain (gr.).....	1	0.041666	0.0020833	0.00228571	0.000142857	0.0648
1 pennyweight (dwt.).....	24	1	0.05	0.0548571	0.00342857	1.5552
1 troy ounce (t. oz.).....	480	20	1	1.0971428	0.0685714	31.104
1 troy pound (t. lb.).....	5,760	240	12	13.165714	0.822857	373.248
1 avoirdupois ounce (av. oz.).....	437.5	18.22.917	0.911458	1	0.0625	28.35
1 avoirdupois pound (av. lb.).....	7,000	291.666	14.58333	16	1	453.6
1 milligram (mg).....	0.015432	0.000643	0.00003215	0.0000035274	0.000002046	0.001
1 gram (g).....	15.432	0.643	0.03215	0.035274	0.0022046	1
1 kilogram (kg).....	15,432	643	32.15	35.274	2.2046	1000

Hundredweight = cwt.

CONVERSION OF WEIGHTS

Because the conversion of avoirdupois, metric and troy weights is so frequently done, the table on page 444, by W. J. Sharwood in Fulton's *A Manual of Fire Assaying*, should be of use.

MENSURATION

Circle:

Area	= diameter squared \times 0.7854 or radius squared \times 3.1416
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Circumference	= diameter \times 3.1416
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Diameter	= circumference \times 0.3183
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Cylinder or prism:

Contents	= area of end \times length or, for a tank, area \times depth
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Surface	= area of both ends + length \times circumference
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Ellipse, area of	= product of two diameters \times 0.7854
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Parabola, area of	= base \times two-thirds altitude
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Parallelogram, area of	= base \times altitude
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Pyramid or cone:*

Contents	= area of base \times one-third altitude
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Surface	= circumference of base \times half of slant height + area of the base
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Sphere:

Contents	= diameter cubed \times 0.5236
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Surface	= diameter \times circumference
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Wedge with parallel

ends, contents of	= area of base \times half altitude
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* May be used in establishing contents of tailing and ore dumps.

TEMPERATURE-CONVERSION FORMULAS AND VARIOUS TEMPERATURES

1. As the centigrade and Fahrenheit temperature scales have 0 and 32° as freezing points and 100 and 212° as boiling points, some method of conversion is necessary, and the following formulas are commonly used:

$$\begin{aligned} C. &= (F. - 32) \times \frac{5}{9} \\ F. &= C. \times \frac{9}{5} + 32 \end{aligned}$$

2. Temperatures obtained by various flames and furnaces are as follows, according to the *Handbook of Chemistry and Physics*:

FLAME AND FURNACE TEMPERATURES

Source	Degrees Centigrade
Industrial furnaces.....	1700 to 1800
Bunsen burner.....	1870
Oxycoal gas flame.....	2000
Oxyhydrogen flame.....	2800
Oxyacetylene flame.....	3500
Electric arc.....	3500

3. Temperatures may be judged approximately by the following color scale:

TEMPERATURE COLOR SCALE

Color	Degrees Centigrade
Incipient red heat.....	500 to 550
Dark-red heat.....	650 to 750
Bright-red heat.....	850 to 950
Yellowish-red heat.....	1050 to 1150
Incipient white heat.....	1250 to 1350
White heat.....	1450 to 1550

FORMULA FOR PULP CONSISTENCY

Let p = percentage solids = weight of solids in unit weight of pulp.

D = dilution = water-solid ratio = parts water by weight per part solids.

S = specific gravity of dry ore.

d = specific gravity of pulp.

Then p/S = volume of solids in unit weight of pulp.

$1 - p$ = volume of water in unit weight of pulp.

$1/d$ = volume of unit weight of pulp.

and $\frac{p}{S} + 1 - p = \frac{1}{d}$

$$d = \frac{S}{p + S(1 - p)} = \frac{S}{S - p(S - 1)}$$

$$d = \frac{D + 1}{D + \frac{1}{S}}$$

$$p = \frac{S(d - 1)}{d(S - 1)} = \frac{1}{D + 1}$$

$$S = \frac{dp}{1 - d(1 - p)} = \frac{d}{1 - D(d - 1)}$$

$$D = \frac{1 - p}{p} = \frac{S - d}{S(d - 1)} = \frac{1}{p} - 1$$

Now, let Z = solid factor = tons solids per fluid ton (32 cu. ft.) of pulp.

G = fluid tons of pulp per ton of dry solids.

q = percentage solids by volume.

Then, as weight of 1 cu. ft. of pulp = $62.5d$, the weight of 1 fluid ton = $32 \times 62.5d$, and

$$\begin{aligned} Z &= \frac{32 \times 62.5d \times p}{2000} = pd \\ &= \frac{d}{D+1} = \frac{pS}{S-p(S-1)} = \frac{S}{DS+1} = \frac{S(d-1)}{S-1} \end{aligned}$$

By definition,

$$G = \frac{1}{Z}$$

$$G = \frac{1}{pd} = \frac{D+1}{d}, \text{ etc.}$$

By definition,

$$q = \frac{2000Z}{62.5 \times 32S} = \frac{Z}{S}$$

$$q = \frac{d-1}{S-1} = \frac{p}{S-p(S-1)}$$

SLIME-DENSITY TABLE

In *Metallurgical and Chemical Engineering* (now *Chemical and Metallurgical Engineering*) for June, 1912, H. B. Lowden presented the following table and explanatory text for slime-density calculations:

Slime-density tables heretofore published have been prepared for use in special cases and are, therefore, not applicable to slimes in which the specific gravity of the solids differs from that for which the table was computed. Their value has been chiefly in indicating convenient forms in which the weight and volume relations may be tabulated for use in the control of the cyanide process. The writer, having experienced the need of a more generally applicable table in his work, has prepared one of considerable range with small intervals, which he feels may be useful to others.

The table is based on the percentage of solid in the slime, opposite which is given the ratio of solid to liquid. The numbers heading the double columns following, are the specific gravities of the dry solid (that of water being taken as unity). The columns headed "S. G." show

SLIME-DENSITY RELATIONS

Per cent solids	Ratio of solids to solution	Specific gravity of pulp and volume of 1 ton in cubic feet, for slimes containing solids of different specific gravities									
		2.50		2.60		2.70		2.80		2.90	
		S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.
5	1:19.000	1031	31.03	1032	31.01	1032	31.01	1033	30.97	1034	30.95
6	1:15.667	1037	30.85	1036	30.82	1039	30.79	1040	30.76	1041	30.74
7	1:13.286	1044	30.66	1045	30.62	1046	30.59	1047	30.56	1048	30.53
8	1:11.500	1050	30.46	1052	30.43	1053	30.39	1055	30.36	1055	30.32
9	1:10.111	1057	30.27	1059	30.23	1060	30.19	1061	30.15	1063	30.11
10	1:9.000	1064	30.08	1065	30.03	1067	29.99	1068	29.95	1070	29.90
11	1:8.091	1071	29.88	1073	29.83	1074	29.79	1076	29.74	1078	29.69
12	1:7.333	1078	29.70	1080	29.64	1082	29.59	1083	29.53	1085	29.48
13	1:6.692	1085	29.50	1087	29.44	1089	29.39	1091	29.33	1093	29.27
14	1:6.144	1092	29.31	1094	29.24	1097	29.19	1099	29.12	1101	29.06
15	1:5.667	1099	29.12	1102	29.05	1104	28.99	1107	28.91	1109	28.85
16	1:5.260	1106	28.93	1109	28.85	1112	28.78	1115	28.71	1117	28.65
17	1:4.882	1114	28.74	1117	28.65	1119	28.58	1123	28.50	1125	28.44
18	1:4.556	1121	28.54	1125	28.45	1128	28.38	1131	28.30	1134	28.23
19	1:4.263	1129	28.35	1133	28.26	1136	28.18	1139	28.09	1142	28.02
20	1:4.000	1136	28.17	1140	28.06	1144	27.98	1147	27.89	1151	27.81
21	1:3.762	1144	27.97	1148	27.87	1152	27.77	1156	27.68	1159	27.60
22	1:3.545	1152	27.78	1157	27.67	1161	27.57	1165	27.47	1168	27.39
23	1:3.348	1160	27.58	1165	27.47	1169	27.37	1174	27.27	1177	27.18
24	1:3.107	1168	27.39	1173	27.27	1178	27.17	1182	27.06	1186	26.97
25	1:3.000	1176	27.21	1182	27.08	1187	26.97	1191	26.85	1196	26.76
26	1:2.846	1185	27.01	1190	26.88	1195	26.77	1201	26.65	1205	26.55
27	1:2.704	1193	26.82	1199	26.68	1205	26.56	1210	26.44	1215	26.34
28	1:2.571	1202	26.62	1209	26.49	1214	26.36	1220	26.24	1224	26.13
29	1:2.448	1211	26.43	1217	26.29	1223	26.16	1229	26.03	1234	25.92
30	1:2.333	1220	26.24	1226	26.10	1233	25.95	1239	25.82	1244	25.71
31	1:2.226	1229	26.05	1236	25.90	1242	25.75	1249	25.63	1255	25.50
32	1:2.125	1238	25.86	1245	25.70	1252	25.55	1259	25.42	1266	25.29
33	1:2.030	1247	25.66	1254	25.50	1262	25.35	1269	25.21	1276	25.08
34	1:1.940	1256	25.47	1264	25.31	1272	25.15	1279	25.01	1287	24.87
35	1:1.857	1266	25.28	1274	25.12	1283	24.95	1290	24.80	1298	24.66
36	1:1.778	1276	25.09	1284	24.91	1293	24.75	1301	24.60	1309	24.45
37	1:1.703	1285	24.90	1295	24.71	1304	24.55	1312	24.39	1320	24.24
38	1:1.632	1295	24.70	1305	24.52	1314	24.35	1323	24.19	1332	24.03
39	1:1.564	1305	24.51	1316	24.32	1326	24.14	1335	23.98	1343	23.82
40	1:1.500	1316	24.32	1326	24.13	1336	23.95	1346	23.77	1356	23.61
41	1:1.439	1326	24.13	1337	23.93	1348	23.74	1357	23.57	1367	23.40
42	1:1.381	1337	23.94	1345	23.73	1359	23.55	1370	23.36	1380	23.19
43	1:1.326	1348	23.74	1359	23.53	1371	23.34	1382	23.16	1392	22.99
44	1:1.273	1359	23.55	1372	23.33	1383	23.15	1395	22.95	1405	22.78
45	1:1.222	1370	23.36	1383	23.14	1395	22.94	1407	22.74	1418	22.57
46	1:1.174	1381	23.17	1395	22.94	1408	22.73	1420	22.54	1432	22.36
47	1:1.128	1393	22.98	1407	22.75	1420	22.54	1433	22.33	1446	22.15
48	1:1.083	1404	22.78	1419	22.55	1433	22.33	1446	22.12	1458	21.94
49	1:1.041	1416	22.59	1431	22.35	1446	22.13	1460	21.92	1473	21.73
50	1:1.000	1429	22.39	1444	22.15	1460	21.92	1473	21.71	1487	21.52
51	1:0.961	1441	22.21	1458	21.96	1473	21.72	1488	21.51	1502	21.31
52	1:0.923	1453	22.02	1471	21.76	1487	21.52	1502	21.30	1517	21.10
53	1:0.887	1466	21.82	1484	21.56	1501	21.32	1516	21.10	1532	20.89
54	1:0.852	1479	21.63	1498	21.36	1515	21.12	1532	20.89	1548	20.68
55	1:0.819	1493	21.44	1512	21.17	1530	20.92	1547	20.69	1564	20.47
56	1:0.786	1506	21.25	1526	20.97	1545	20.72	1563	20.48	1580	20.26
57	1:0.754	1520	21.06	1540	20.77	1560	20.51	1579	20.27	1596	20.05
58	1:0.724	1534	20.86	1555	20.58	1574	20.31	1595	20.07	1613	19.84
59	1:0.695	1548	20.67	1572	20.38	1591	20.11	1611	19.86	1629	19.63
60	1:0.667	1563	20.48	1585	20.18	1607	19.91	1628	19.66	1645	19.42
61	1:0.639	1577	20.29	1601	19.98	1623	19.71	1645	19.45	1664	19.21
62	1:0.613	1592	20.10	1617	19.79	1641	19.51	1662	19.25	1683	19.00
63	1:0.587	1608	19.90	1633	19.59	1657	19.30	1681	19.04	1703	18.79
64	1:0.563	1623	19.71	1650	19.40	1675	19.10	1698	18.84	1723	18.58
65	1:0.538	1639	19.52	1667	19.20	1692	18.90	1718	18.63	1742	18.37
66	1:0.515	1656	19.32	1684	19.00	1711	18.70	1738	18.43	1762	18.16
67	1:0.493	1672	19.14	1701	18.80	1730	18.50	1757	18.22	1783	17.95
68	1:0.471	1689	18.94	1719	18.61	1749	18.30	1776	18.01	1803	17.74
69	1:0.449	1706	18.75	1738	18.41	1768	18.10	1797	17.81	1825	17.53
70	1:0.429	1724	18.56	1757	18.21	1786	17.90	1818	17.60	1847	17.32

SLIME-DENSITY RELATIONS.—(Continued)

Per cent solids	Ratio of solids to solution	Specific gravity of pulp and volume of 1 ton in cubic feet, for slimes containing solids of different specific gravities									
		3.00		3.10		3.20		3.30		4.50*	
		S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.	S.G.	Vol.
5	1:19.000	1035	30.93	1035	30.92	1036	30.90	1036	30.89	1040	30.76
6	1:15.667	1042	30.72	1042	30.70	1043	30.68	1043	30.66	1049	30.51
7	1:13.286	1049	30.51	1049	30.48	1050	30.46	1051	30.44	1058	30.26
8	1:11.500	1056	30.30	1057	30.27	1058	30.24	1059	30.21	1067	30.01
9	1:10.111	1064	30.09	1065	30.05	1066	30.02	1067	29.99	1075	29.76
10	1: 9.000	1071	29.87	1072	29.83	1074	29.80	1075	29.77	1084	29.51
11	1: 8.091	1078	29.65	1080	29.61	1082	29.58	1083	29.54	1093	29.26
12	1: 7.333	1087	29.44	1088	29.40	1090	29.36	1091	29.32	1102	29.01
13	1: 6.692	1095	29.23	1096	29.18	1098	29.14	1099	29.10	1112	28.76
14	1: 6.144	1103	29.01	1105	28.96	1106	28.92	1108	28.88	1122	28.52
15	1: 5.667	1111	28.80	1113	28.74	1115	28.70	1117	28.66	1132	28.27
16	1: 5.250	1119	28.59	1122	28.53	1124	28.45	1125	28.43	1142	28.02
17	1: 4.882	1128	28.37	1130	28.31	1132	28.26	1134	28.21	1152	27.77
18	1: 4.556	1136	28.16	1139	28.10	1141	28.04	1143	27.99	1163	27.52
19	1: 4.263	1145	27.95	1148	27.88	1150	27.82	1153	27.76	1173	27.27
20	1: 4.000	1154	27.73	1157	27.66	1159	27.60	1162	27.54	1184	27.02
21	1: 3.762	1163	27.52	1166	27.44	1169	27.38	1171	27.32	1194	26.77
22	1: 3.545	1172	27.31	1175	27.23	1178	27.16	1181	27.09	1206	26.52
23	1: 3.348	1181	27.09	1184	27.01	1188	26.94	1191	26.87	1218	26.28
24	1: 3.167	1190	26.88	1194	26.79	1198	26.72	1201	26.65	1230	26.03
25	1: 3.000	1200	26.67	1204	26.58	1208	26.50	1211	26.42	1241	27.78
26	1: 2.846	1210	26.45	1214	26.37	1218	26.28	1222	26.20	1253	26.53
27	1: 2.704	1220	26.24	1224	26.15	1228	26.06	1232	25.98	1266	26.28
28	1: 2.571	1230	26.03	1234	25.93	1239	25.84	1242	25.75	1278	25.03
29	1: 2.448	1240	25.81	1244	25.71	1249	25.62	1253	25.53	1291	24.78
30	1: 2.333	1250	25.60	1255	25.50	1260	25.40	1264	25.31	1304	24.53
31	1: 2.226	1261	25.39	1266	25.28	1271	25.18	1275	25.08	1317	24.28
32	1: 2.125	1271	25.17	1277	25.06	1282	24.96	1287	24.86	1331	24.04
33	1: 2.030	1282	24.96	1288	24.85	1293	24.74	1299	24.64	1345	23.79
34	1: 1.940	1293	24.75	1299	24.63	1305	24.52	1311	24.41	1359	23.54
35	1: 1.857	1304	24.53	1310	24.41	1317	24.30	1323	24.19	1374	23.29
36	1: 1.778	1316	24.32	1322	24.19	1329	24.08	1335	23.97	1389	23.04
37	1: 1.703	1328	24.11	1334	23.98	1341	23.80	1347	23.75	1404	22.79
38	1: 1.632	1340	23.89	1346	23.76	1353	23.64	1360	23.52	1420	22.54
39	1: 1.564	1351	23.68	1358	23.55	1366	23.42	1373	23.30	1435	22.29
40	1: 1.500	1363	23.47	1371	23.33	1379	23.20	1387	23.08	1461	22.04
41	1: 1.439	1376	23.26	1384	23.11	1393	22.98	1400	22.85	1468	21.79
42	1: 1.381	1389	23.04	1397	22.89	1406	22.76	1414	22.63	1485	21.55
43	1: 1.326	1402	22.83	1411	22.68	1419	22.54	1428	22.41	1502	21.30
44	1: 1.273	1415	22.61	1425	22.46	1433	22.32	1442	22.18	1519	21.05
45	1: 1.222	1429	22.40	1438	22.24	1447	22.10	1456	21.96	1538	20.80
46	1: 1.174	1443	22.19	1452	22.02	1462	21.88	1471	21.74	1557	20.55
47	1: 1.128	1457	21.97	1467	21.81	1477	21.66	1487	21.51	1576	20.30
48	1: 1.083	1471	21.76	1483	21.60	1493	21.44	1503	21.29	1595	20.08
49	1: 1.041	1485	21.55	1497	21.38	1508	21.22	1519	21.07	1615	19.81
50	1: 1.000	1500	21.33	1512	21.16	1524	21.00	1535	20.85	1637	19.56
51	1: 0.961	1515	21.12	1528	20.94	1540	20.78	1551	20.62	1588	19.31
52	1: 0.923	1531	20.91	1544	20.73	1556	20.58	1568	20.40	1679	19.06
53	1: 0.887	1547	20.69	1560	20.51	1573	20.34	1585	20.18	1700	18.81
54	1: 0.852	1563	20.48	1577	20.29	1590	20.12	1603	19.96	1724	18.56
55	1: 0.819	1579	20.27	1594	20.08	1608	19.90	1621	19.73	1748	18.31
56	1: 0.786	1596	20.05	1611	19.87	1626	19.68	1640	19.51	1772	18.06
57	1: 0.754	1613	19.84	1628	19.65	1645	19.46	1659	19.29	1796	17.81
58	1: 0.724	1631	19.63	1646	19.43	1663	19.24	1678	19.06	1822	17.56
59	1: 0.695	1649	19.41	1665	19.21	1682	19.02	1697	18.84	1848	17.32
60	1: 0.667	1667	19.20	1684	19.00	1702	18.80	1718	18.62	1875	17.07
61	1: 0.639	1686	18.99	1704	18.78	1722	18.58	1739	18.39	1903	16.82
62	1: 0.613	1705	18.77	1724	18.56	1742	18.36	1761	18.17	1932	16.57
63	1: 0.587	1724	18.56	1745	18.34	1764	18.14	1783	17.95	1961	16.32
64	1: 0.563	1745	18.35	1766	18.12	1786	17.92	1805	17.72	1992	16.07
65	1: 0.538	1765	18.13	1786	17.91	1808	17.70	1828	17.50	2023	15.82
66	1: 0.515	1786	17.92	1808	17.69	1830	17.48	1852	17.28	2054	15.57
67	1: 0.493	1808	17.71	1831	17.47	1853	17.26	1876	17.06	2088	15.32
68	1: 0.471	1830	17.49	1854	17.26	1877	17.04	1901	16.88	2123	15.08
69	1: 0.449	1852	17.28	1878	17.04	1902	16.82	1927	16.61	2159	14.83
70	1: 0.429	1875	17.07	1902	16.83	1926	16.60	1953	16.39	2195	14.58

* 80 per cent pyrite and 20 per cent quartz.

the specific gravities of the slime, that of water being taken as 1000; that is, the figures show directly the weight of a liter of slime in grams. The columns headed "Vol." show the number of cubic feet of the slime in 1 ton of 2000 lb.

The specific gravities of solids chosen will probably cover the range of slimes ordinarily met with, and the intervals are sufficiently small to admit of interpolation without appreciable error. The last column (4.50) is a hypothetical concentrate and is the specific gravity of a mixture of 80 per cent pyrite and 20 per cent quartz. The average specific gravity of working cyanide solutions is so small as to be negligible.

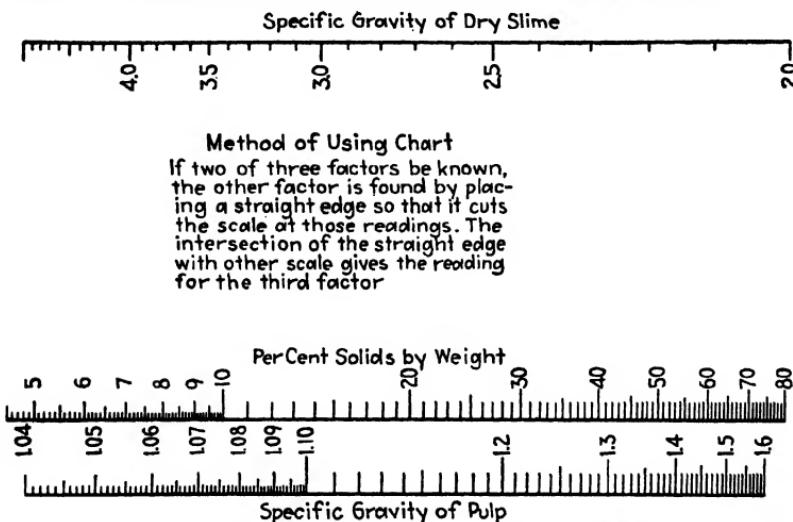


FIG. 133.—Scale for slime-density calculations.

The table is convenient for ascertaining the amount of solid and of solution in slime pulps from the number of cubic feet, determined by rod or float, in the tank; and specific gravity of the slime, determined by taking the weight of a liter or by a specific gravity indicator in the tank. It is useful in calculations for ascertaining the amount of solution to be abstracted or added in thickening and diluting, for correcting the strength of the solutions, for checking tonnage and for other purposes.

Assume that in a plant in which the specific gravity of the solid is 2.7, a tank is shown, by the depth of pulp in it, to contain 3530 cu. ft. of pulp, a liter of which weighs 1223 grams. From the table it is found that the specific gravity 1223 corresponds to 26.16 cu. ft. per ton and to 29 per cent solid. The weight of pulp, therefore, is

$$3530 \div 26.16 = 135 \text{ tons},$$

and the weight of solids $135 \times 0.29 = 39.15$ tons. The weight of solution is, by difference, 95.85 tons. If the solution titrates 1.05 lb. cyanide per ton and it is desired to bring the strength up to 2.5 lb. per ton, we have $2.5 - 1.05 = 1.45$ lb. cyanide to be added per ton. Therefore $95.85 \times 1.45 = 139$ lb. cyanide to be added to the tank.

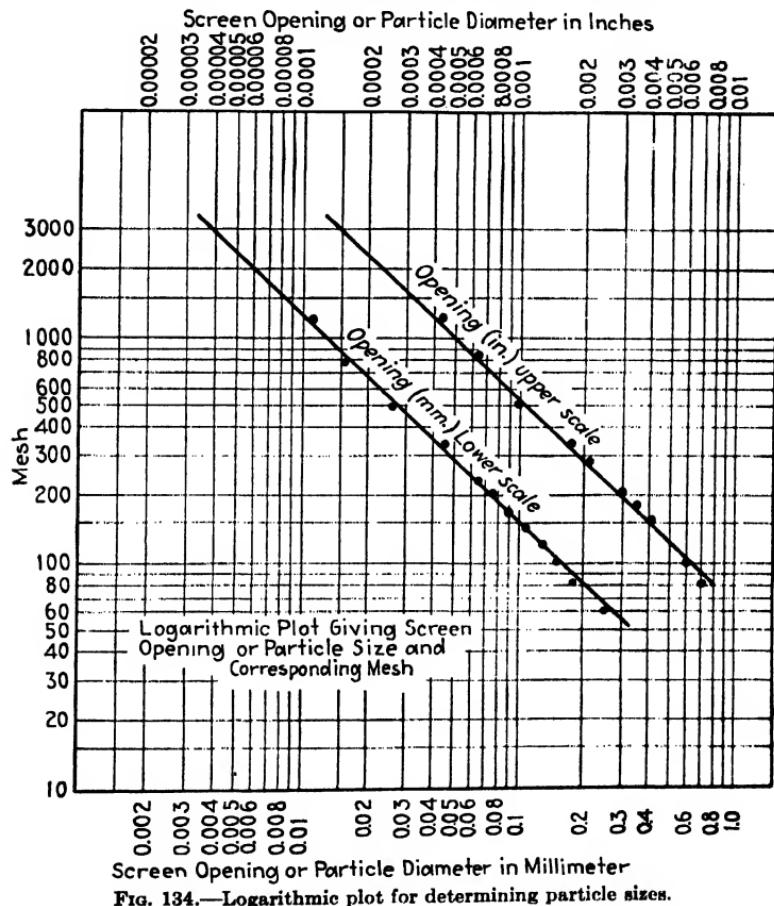


FIG. 134.—Logarithmic plot for determining particle sizes.

The table is useful in determining the sizes of tanks necessary for any given capacities. Thus, if it is desired to agitate 50 tons of dry slime (specific gravity of solid 2.6) with three parts solution, the table shows this to contain 25 per cent solids and to have a volume of 27.08 cu. ft. per ton; therefore $50 \div 0.25 = 200$ tons of slime $\times 27.08 = 5416$ cu. ft., the required effective working capacity of the tank, to which an amount must be added to secure the desired height of curb above the charge.

The preceding logarithmic plot (Fig. 134) is a convenient means for determining particle sizes in inches or millimeters for any screen mesh. (Tyler Standard Scale — $\sqrt{2}$.) The meshes are in

SIEVE SERIES, TYLER STANDARD SCREEN SCALE AND I.M.M. SERIES

Mesh		Aperture, inches		Aperture, millimeters	
Tyler	I.M.M.	Tyler	I.M.M.	Tyler	I.M.M.
8	5	0.093	0.1	2.362	2.540
10	8	0.065	0.062	1.651	1.574
14	12	0.046	0.0416	1.168	1.056
20	16	0.0328	0.0312	0.833	0.792
28	20	0.0232	0.025	0.589	0.635
35	30	0.0164	0.0166	0.417	0.421
48	40	0.0116	0.0125	0.295	0.317
65	60	0.0082	0.0083	0.208	0.211
100	100	0.0058	0.005	0.147	0.127
150	120	0.0041	0.0033	0.104	0.084
200	150	0.0029	0.0025	0.074	0.063

even figures, and the standard meshes are obtained by interpolation, *i.e.*, 28, 35, 48, 65, etc.

Micron sizes are readily obtained from the millimeter scale by changing the decimal point. (1 millimeter = 1000 microns.)

BOOKS ON CYANIDATION

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FIG. 135.—Bullion pour at Homestake.

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